Energy, Mines and Resources Canada

Énergie, Mines et Ressources Canada

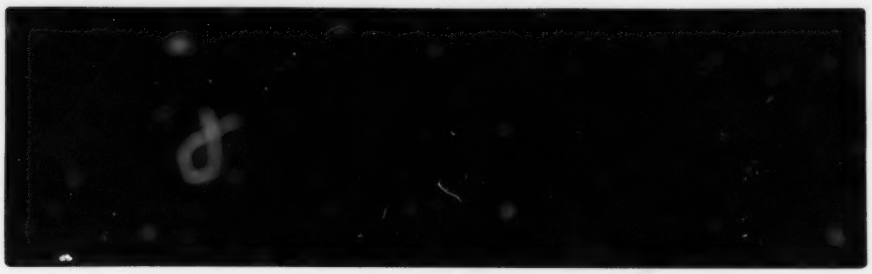
CANMET

Canada Centre for Mineral and Energy Technology

Centre canadien de la technologie des minéraux et de l'énergie

Mineral Sciences Laboratories

Laboratoires des sciences minérales



Canada

8-9259

A Comminution Study for Reduction of Grinding Energy Cost for Granulated Copper Slag

Final Report
by Muammer Kaya and Robert E. McIvor

Grinding Process Development (GPD) Co. Ltd.
Montréal, Quebec

Work on the project was conducted under the auspices of
the Canada Centre for Mineral and Energy Technology (CANMET),
Energy, Mines and Resources Canada

DSS File No.: 007SQ.23440-8-9259
DSS Contract Serial No.: 23440-8-9259/01-SQ
CANMET Project No.: 303705
CANMET Scientific Authority: Dr. Esther Douglas
Date: October 1989

68 pp

MICROMEDIA

TABLE OF CONTENTS

	Page
Executive Summary	i
Resume pour la Direction	ii
Introduction: Objectives and General Approach	1-2
Roller Press and Zeisel Grindibility Tests at KHD in W. Germany	2-4
Grindibility Tests Performed at McGill	4-11
-Experimental	
-Description of the Sample	4
-Size Analysis of the Test Feed	5-6
-Density of the Feed	6
-Mill Solids Load	6-7
-Calculation of the Ideal Circulating Load	8
-Calculation of the Ideal Potential Product	8
-Calculation of the Number of Revolutions	8
-Bond Work Index Calculations	8-9
-Grindibility Tests	9-11
-Closed Circuit Grinding	9
-Open Circuit Grinding	10
-Product Generation	10-11
-Results	11-27
-Grindibility Tests	11-17
-Product Generation	18-20
-Interpretation of the Test Results	20-27
-Industrial Mill Design Alternatives	20-24
-Closed versus Open Circuit Ball Milling	24-27
Conclusions	28
References	29
Appendices	
-Test Report from KHD	
-Zeisel Grindibility Test Procedure	
-Standard Test Method for Fineness of Portland Cement by Air Permeability Apparatus	

EXECUTIVE SUMMARY

The objective of this work was to investigate the potential for reducing the total energy required to grind Kidd Creek Mines granulated copper slag to approximately 3000 Blaine fineness for use in cemented backfill. The effect of high pressure roll crushing prior to ball milling was considered as the primary alternative to ball mill grinding alone. Closed versus open circuit ball milling was also considered.

Samples of granulated copper slag were sent to KHD in Cologne, Germany, where they underwent roll crushing at 60 and 80 bar pressures. The original "as received" and roll crushed products were also tested for their ball milling grindability characteristics in their Zeisel grindability tests. The results indicated that on a commercial scale, total energy consumption to produce a 3000 Blaine product fineness could be reduced from 63 kWh/mt estimated by the Zeisel test to 53.5 kWh/mt by roll crushing at 80 bar prior to ball milling. On a similar basis, Bond ball mill work index tests at McGill indicated that total energy consumption could be reduced from 73.8 to 53.2 kWh/mt. It has to be noted that total energy consumption to grind Kidd Creek Mines granulated copper slag to 3000 Blaine by ball milling has been previously estimated by CANMET and by Kidd Creek Mines as 78 and 81.5 kWh/mt, respectively.

Although the recovery of suitably crushed material in a commercial roll crusher should be 90 to 95%, it was only about 35% on this material unit during the first tests at KHD. Because more material was needed to perform the grinding tests at McGill and to prepare sufficient mass (approximately 50 kg) for subsequent testing of cementitious properties, a second shipment had to be sent to KHD for roll crushing at 80 bar.

Open circuit (single batch) grinding tests in the Bond apparatus surprisingly showed that in terms of the production of material of a given Blaine fineness, open circuit grinding is much more efficient than closed circuit grinding. However, if the criteria for "efficiency" is taken as the production rate of new product size material closed circuit grinding is then more efficient.

Finally, a 420 mm batch mill was operated in closed circuit with screens to produce approximately 75 kg of material with a fineness of 3270 Blaine. This material was shipped to CANMET for subsequent testing of its cementitious properties.

RESUME POUR LA DIRECTION

L'objectif de ce travail était d'étudier le potentiel de diminution d'énergie totale requise pour broyer des scories de cuivre granuleuses de Kidd Creek Mines. L'objectif de finesse était de 3000 Blaine pour que le produit puisse être utilisé dans les remblayages cimentés. L'effet du concasseur à cylindres à haute pression sur les scories avant le broyage par boulets semblait être l'alternative préférée au simple broyage par boulets. Le broyage en circuit fermé au lieu d'ouvert entra en considération.

Des échantillons de scories granuleuses furent expédiés à KHD à Cologne en Allemagne. Là, ils furent concassés dans le concasseur à cylindres à des pressions de 60 et 80 bars. L'échantillon original et les produits du concasseur furent soumis au test de broyabilité de Zeisel. Ce test mesure les caractéristiques de broyabilité de matériaux au niveau du broyage par boulets. Sur une échelle commerciale, les résultats du test de Zeisel indiquèrent que la consummation d'énergie totale pouvait être réduite de 63 à 53.5 kWh/tm pour obtenir un matériel de 3000 Blaine lorsque les scories étaient concassées à une pression de 80 bars avant le broyage par boulets.

Les résultats des tests de broyabilité de Bond au niveau du broyage par boulets indiquèrent que la consommation totale d'énergie pouvait être réduite de 73.8 à 53.2 kWh/tm. Ces résultats s'appliquent aussi au matériel concassé à 80 bars comparé à l'échantillon original. Le CANMET et Kidd Creek Mines avaient estimé que la consommation totale d'énergie requise pour broyer les scories de cuivres granuleuses de Kidd Creek Mines à une finesse de 3000 Blaine était de 78 et 81.5 kWh/mt respectivement.

La récupération de matériel de finesse désirée d'un concasseur à cylindres est généralement de 90 à 95% à l'échelle commerciale. Lors du traitement des scories à KHD, la récupération de matériel ne fut que de 35%. Ceci ne produisit pas assez de matériel pour les tests restant à entreprendre; un second échantillon de scories fut expédié à KHD et concassé à 80 bars de pression.

Le broyage en circuit ouvert (par lots) dans le broyeur à boulets utilisé pour les tests de Bond se démontra plus efficace que le broyage en circuit fermé en terme de production de matériel d'une finesse de Blaine désirée. Cependant, si le critère "d'efficacité" est le taux de production de matériel de finesse désirée, alors le broyage en circuit fermé est plus efficace.

Finalement, un broyeur de 420 mm de diamètre (en circuit fermé avec tamis) servit à produire 75 kg de matériel de finesse 3270 Blaine. Ce matériel fut expédié au CANMET pour des tests se reliant à ses propriétés de cimentation.

INTRODUCTION

OBJECTIVES AND GENERAL APPROACH

Batch grinding experiments have shown that Kidd Creek Mines granulated copper slag has an extremely high energy requirement for size reduction to develop a surface area suitable for use as a replacement for portland cement in cemented backfill (i.e. 78 kWh/mt for 3000 Blaine and 240 kWh/mt for 4000 Blaine). These tests were conducted by grinding in a test ball mill until desired Blaine fineness was achieved (1)

The purpose of this work was to investigate the potential for reduction in the overall energy requirement to achieve a product with approximately 3000 Blaine fineness. Two alternatives to the single step batch process were considered. The first, and the primary subject of this investigation, was the use of high pressure roll crushing prior to ball milling. The second, and a secondary objective of this investigation, was ball milling in closed circuit with a size classification device.

Both of these steps may produce substantial reduction in overall energy requirements to achieve the desired product fineness. Previous laboratory tests have shown an expected energy savings of over 20% by introducing the high pressure roll crushing stage before the ball mill circuit on a blast furnace slag (2). The advantages of closed versus open circuit ball milling have also been widely demonstrated in both theory and practice (3). As expected, both of these steps have been incorporated into full scale plant practice (4).

The general approach taken was to estimate the total commercial energy requirement with and without the roll crushing stage using a laboratory scale roll crusher and the Bond ball mill work index test apparatus (5) in its normal closed circuit mode. Batch grinding in the Bond apparatus was also carried out to

somewhat larger test mill was operated in closed circuit to produce the desired quantity of product for subsequent testing of cementitious properties.

Three sample lots, one of 50 kg, one of 120 kg, and another 300 kg, were received from CANMET and Kidd Creek Mines at different times. The 120 and 300 kg samples were directly shipped to the Humboldt-Wedag KHD laboratory in Cologne, W. Germany for roll crushing. The 50 kg sample was used at McGill University for Bond work index tests. 16 kg of product (i.e. 8 kg at 80 bars and 8 kg at 60 bars) from the first shipment and 155 kg from the second shipment were received back from KHD for grindibility tests at McGill.

ROLLER PRESS AND ZEISEL GRINDIBILITY TESTS AT KHD IN W. GERMANY

The first shipment (120 kg) sample was roller pressed at two pressures (i.e. 60 and 80 bars) using a laboratory roller press type RP 80-12. The flakes obtained from each tests were disagglomerated in a hammer mill.

The particle size distributions of the original material and the roll-crushed disagglomerated flakes were determined and Zeisel grindibility tests were performed. Table 1 shows the test conditions. A copy of the original KHD report can be found in Appendix 1 along with subsequent correspondence. The details of the Zeisel grindibility test are given in Appendix 2.

Table 1: Roller press tests conditions for the first shipment.

	TEST 1	TEST 2
Roller press pressure	80 bar	60 bar
Amount of sample used	36 kg	30 kg
Standard pressure	94 kN/cm	71 kN/cm
Flake thickness	7-8 cm	7-9 cm
Flake width	90 mm	90 mm
Flake density	3.11 g/cm^3	3.06 g/cm^3
Mass recovery	38%	30%
Nipping characteristics	moderately good	worse than Test 1

The grindibility of the original sample and the dis-agglomerated flakes from both tests was determined according to the dry open-circuit principle with the testing equipment of Messrs. Tonindustrie, Berlin (Zeiseltest). About 60 g of each sample in a particle size range between 0.75 mm and 1 mm was ground in 7 or 8 steps to 3000 Blaine. The specific energy consumptions from the Zeisel tests are given in Table 2. According to KHD, these results may be scaled directly to a commercial size flowsheet (see Appendix 1). Size distributions of the products from the Zeisel grindibility tests were determined using a Cilas Lasergranulameter at KHD (Table 3). The second shipment (300 kg) sample was roll crushed at 80 bar only. No additional Zeisel grindibility tests were performed.

Table 2: Comparative energy requirements for grindibility to 3000 Blaine from KHD roller press and Zeisel grindibility tests.

Energy Consumption (kWh/mt)	Without Roll Crushing	With Roll Crushing	
		80 Bar	60 bar
Roll Crushing (approximate)	-	5.5	5.5
Ball Milling	63	48.0	52.0
TOTAL	63	53.5	57.5

Note that the mass recovery of crushed product from the first pair of roll crushing tests (at 80 and 60 bar) was only about 30-38%. This necessitated the shipment of the second sample to KHD. This time, the mass recovery was 52%. It has been pointed out by KHD that the mass recovery of a commercial roller press would be much higher, in the order of 90-95%.

Table 3: Particle size distributions of the Zeisel grindibility test products.

Particle Size (μm)	Cumulative Percent Retained		
	Original (as received)	Test 1 (80 bar)	Test 2 (60 Bar)
1.000	95.0	94.1	94.3
1.500	93.8	92.4	92.6
2.000	90.6	88.2	88.4
3.000	85.6	82.3	82.5
4.000	79.7	76.5	76.8
6.000	70.3	68.1	68.8
8.000	61.0	60.2	61.1
12.00	48.6	48.6	49.8
16.00	39.1	40.0	41.0
24.00	28.5	28.9	29.8
32.00	20.3	21.3	22.0
48.00	10.1	9.7	10.2
64.00	5.9	6.6	6.9
96.00	0.9	0.0	0.5
128.0	0.0	0.0	0.0
192.0	0.0	0.0	0.0
1000	0.0	0.0	0.0

GRINDIBILITY TESTS PERFORMED AT McGILL

EXPERIMENTAL

Grindibility tests were carried out in the standard Bond ball mill apparatus (Table 4) on the slag as received from CANMET and on the 80 Bar roll-crushed product from KHD.

DESCRIPTION OF THE SAMPLE

The granulated copper slag is a black colored material. The particles are mainly regular shaped, partly rounded and edged to sharp edged. They appear glassy. The moisture content of the sample was around 2.6% by mass.

Table 4: Standard Bond work index mill characteristics for grindibility tests.

Mill Dimensions (LxW)	: 30.48x30.48 cm
Mill Speed (empty)	: 70 rpm
Lifter	: smooth
Ball Load	: 20.125 kg
Ball Size Distribution	Number
3.68 cm	: 43
2.97 cm	: 67
2.54 cm	: 10
1.91 cm	: 71
1.55 cm	: 94

SIZE ANALYSES OF THE TEST FEED

Samples were first dried in an oven overnight at 100^{0} C and then split into subsamples using a spinning riffle. The size distribution of the test feed is required to determine the percentage of material which passes the test control size of 38 μm and the F80 of the Bond test feed. Several subsamples were wet screened on this screen. After drying the oversize and undersize fractions, the former were dry screened for 30 minutes to determine the feed size distribution. The results were averaged. Table 5 shows the size distribution of the original (i.e. 50 kg sample as received from CANMET) and the roll-crushed products from the two shipments that were crushed at 80 Bar by KHD.

The original (as received) sample mainly contains particles between 1700 μm (8 mesh) and 300 μm (48 mesh). The fraction of particles finer than 212 μm (65 mesh) is less than 5% of the feed. The roll crushed material has a wider particle size distribution. More than 20% of the feed is finer than 38 μm (400 mesh). Size distributions of the two roll-crushed materials were very similar. Figure 1 compares the size distributions of the original sample as received and screen analyzed at McGill and KHD. It appears that the first shipment for KHD did not have exactly the same size distribution as the McGill sample. Note that check of the size analysis reported by KHD on the first shipment of roll crushed

product showed no discrepancy between KHD and McGill sizing techniques.

TABLE 5: Size distributions of the sample as received from CANMET and the 80 Bar roll-crushed products from KHD.

Screen opening size		Original Sample		Roll-Crushed Sample			
				First Shipment		Second Shipment	
(mesh)	(μm)	Ind. (%)	Cum. Pass. (%)	Ind. (%)	Cum. Pass. (%)	Ind. (%)	Cum. Pass. (%)
6	2360	0.65	99.35	0.00	100.00	0.00	100.00
8	1700	0.52	98.83	0.05	99.95	0.04	99.96
14	1180	9.11	89.73	0.36	99.59	0.09	98.87
20	850	20.98	68.74	2.37	97.22	2.84	96.03
28	600	30.76	37.98	8.44	88.78	8.67	87.36
35	425	20.88	17.10	11.83	76.95	10.84	76.52
48	300	8.99	8.11	11.39	65.56	10.07	66.45
65	212	4.02	4.09	9.99	55.57	9.82	56.63
100	150	2.05	2.04	11.14	44.43	10.80	45.83
150	106	1.02	1.01	8.72	35.71	8.73	37.10
200	75	0.48	0.53	7.97	27.74	9.76	27.34
270	53	0.20	0.33	4.88	22.86	2.70	24.64
325	45	0.05	0.29	0.53	22.33	1.17	23.47
400	38	0.02	0.27	1.49	20.84	1.80	21.67
-400	-38	0.27	-	20.84	-	21.67	-

DENSITY OF THE FEED

Feed density was determined on the as received sample using a weight displacement method. Two subsamples were used for density measurement and results were averaged out. The test sample density was 3.7 g/cm^3.

MILL SOLIDS LOAD

The Bond ball mill work index test requires 700 cm^3 of material, which was equal to 1422 grams. The mill solids load was the same for both the open and closed circuit grinding tests.

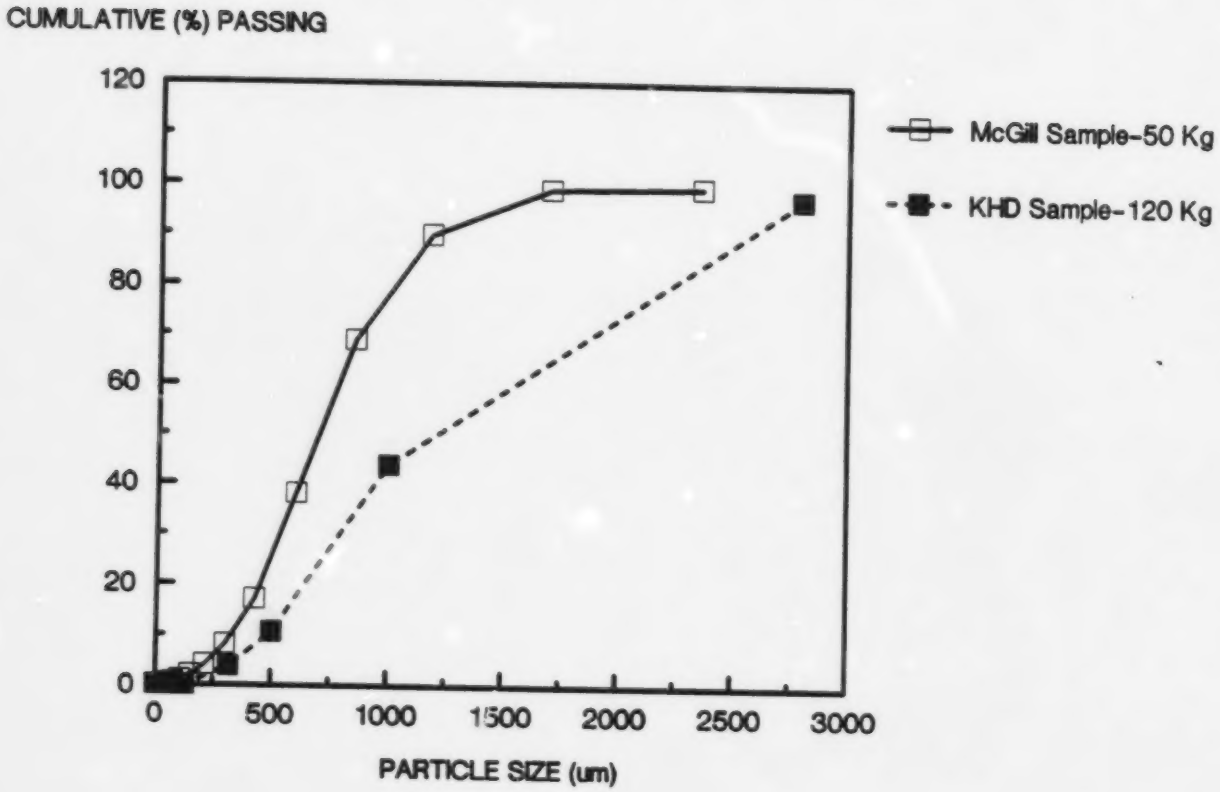


Figure 1: Size distributions of the original 50 kg sample shipped to McGill and 120 kg sample shipped to KHD.

CALCULATION OF THE IDEAL CIRCULATING LOAD

The Bond ball mill work index test calls for a 250% circulating load ratio. The ideal circulating load (i.e. recirculation of coarse solids) can be calculated from:

$$\text{Ideal Circulating Load (g)} = \text{Mill Solids Load} - \frac{\text{Mill Solids Load}}{1 + \frac{250\ \%}{100\%}}$$

CALCULATION OF THE IDEAL POTENTIAL PRODUCT

The ideal potential product is the mill load minus the ideal circulating load:

$$\text{Ideal Potential Product} = \text{Mill Solids Load} - \text{Ideal Circulating Load}$$

CALCULATION OF THE NUMBER OF REVOLUTIONS

The number of revolutions for cycle 1 was fixed at 200. The number of revolutions for cycle 2 and subsequent cycles of the test were estimated as follows:

$$\text{Number of Rev.} = \frac{\text{Ideal Potential Prod. (g)} - \text{Weight of Prod. in Test Feed (g)}}{\text{Production Rate of Product in the Previous Cycle (g/rev)}}$$

BOND WORK INDEX CALCULATIONS

Work index, W_i, is the calculated kWh per ton applied in reducing material of infinite particle size to 80% passing 100 μm (equivalent to approximately 65% passing 200 mesh).

$$W_i = \frac{44.5}{(P1)^{0.23} \cdot (GPR)^{0.82} \cdot \left[\left[\frac{10}{P^{0.5}}\right] - \left[\frac{10}{F^{0.5}}\right]\right]} \qquad \text{Eq. 1}$$

where Wi : work index in short ton
P1 : screen cut size in micron
GPR: grams per revolution
F : 80% passing size of the Bond test circuit feed in micron
P : 80% passing size of the Bond test circuit product in micron

Specific power consumption, W, can be calculated from:

$$W = 10.W_i \cdot \left[\frac{1}{P_{80}^{0.5}}\right] - \left[\frac{1}{F_{80}^{0.5}}\right] \qquad \text{Eq.2}$$

where P_{80} and F_{80} are the 80% passing sizes of the commercial circuit feed and product, respectively.

GRINDIBILITY TESTS

Closed Circuit Grinding: In closed circuit grinding, the product was removed by wet screening between each cycle and fresh feed was combined with the circulating load for the next cycle. At first, the screen size was 45 μm (325 mesh). This did not achieve the desired fineness of 3000 Blaine in the test product. It was replaced with a 38 μm (400 mesh) screen, which is the finest screen used for Bond work index testing. The initial mill load was 1422 g, 0.27% of which was finer than 38 μm. The material was dry ground for 200 revolutions and the discharge was wet screened. Both undersize and oversize fractions were dried in the oven and then, the coarse fraction was dry screened for 30 min. The product was discarded and an equivalent amount of test feed was combined with the circulating load for the next cycle. This procedure was repeated until the equilibrium (i.e. constant grams/revolution and 250% recirculation ratio) was reached. The product (-38μm) size distribution was determined using Sedigraph 5000D. The fineness of each product after the third cycle was determined using a Blaine air permeability apparatus. Standard ASTM C204-84 test method was followed for Blaine fineness measurements (Appendix 3).

Open Circuit Grinding: In the open circuit grinding, the same amount of mill load (1422 g) was ground. At certain time intervals, the mill was stopped and the product fineness was checked. Grinding was continued until 3000 Blaine was achieved.

PRODUCT GENERATION

Due to the fixed grinding time in a batch mill versus the residence time distribution due to mixing in a continuous mill and the high efficiency of wet screening in the Bond locked-cycle test versus the performance of an actual commercial air classifier, the product size distribution from the Bond test is not representative of a continuous commercial circuit. Therefore, in the final step, a slightly larger mill (described in Table 6) was used in closed circuit with dry screening at 53 μm (200 mesh) to produce a composite product of a Blaine fineness of 3000 cm^2/g (or finer), which is suitable for subsequent testing of cementitious properties. This material was subsequently shipped to CANMET in Ottawa. Table 7 shows the ball load used in the mill.

Table 6: Mill characteristics for the product generation.

Mill Dimensions (L*W)	: 42*41 cm
Mill Speed (empty)	: 30 rpm
Mill Motor Power	: 1 HP
Mill Load	: 15 kg slag
Lifters	: present
Ball Charge	: 75 kg
Ball Size	: 1.27-5.08 cm
Grinding Time	: 5 hrs
Screen Size	: 53 μm (200 mesh)
Screening Time	: 5 min
Number of Screens	: 8
Screen Load	: 300 g

Table 7: Grinding ball size distribution for the product generation.

Size (cm)	Number	Weight min. (gram)	Weight max. (gram)	Weight ave. (gram)	Total Weight (gram)
1.27	179	32	35	33.85	6059
1.91	68	42	58	54.70	3720
2.54	692	68	80	78.35	54230
3.18	26	120	140	130.08	3382
3.81	21	215	230	220.81	4637
5.08	4	735	750	743.00	2972
TOTAL					75000

This final step required adjustments of grinding and screening times to achieve the necessary product fineness. During cycle 1, the grinding time changed from 1 to 6 hrs, and the screening time from 5 to 10 minutes. First, one batch of sample was ground for different times. At the end of the grinding period, the mill was discharged and screened. The overall and size-by-size product finenesses were measured. Then, the product and oversize were combined and returned to the mill for the next cycle until reaching the necessary product fineness. This determined that a five-hour grinding time and five-minute screening time would provide the desired product fineness. Product was then generated in 8 batches at five-hour grinding and five-minute screening.

RESULTS

GRINDIBILITY TESTS

Table 8 summarizes the Bond work index test results for the as received sample and the roll-crushed material (80 bar). Eleven cycles of grinding for the as received sample and nine cycles for the roll-crushed material were carried out. The average rate of production in the last three cycles were 0.371 and 0.643 g/rev, respectively. Corresponding Blaine finenesses of the Bond test products (-38 μm) were 2285 and 2101 cm^2/g, and 80% product passing sizes were 33 and 35 μm. Grinding periods at equilibrium

Material: Granulated Copper Slag from CANMET (As Received)
Mill Solids Load: 1422 grams (700 cc)
Ideal Potential Product: 1422/3.5= 406.3
Ideal Circulating Load: 1422-406= 1015.7
Product Fraction in the Test Feed: 0.27%
Test Control Size: 38 μm (400 mesh)
Product 80% Passing Size: 33 μm
Feed 80% Passing Size: 1027 μm

	Cycle Number	New test feed add. (g)	Number Rev.	Mass of prod. in new test feed (g)	Mass of prod. in mill disch. (g)	Net mass of prod. (g)	Prod. rate (g/rev)	Circ. Load ratio (%)	Blaine (cm2/g)
	1	1422.00	200	3.84	38.00	34.16	0.171	4149	
	2	38.00	2378	0.10	570.00	569.90	0.240	149	
	3	569.90	1689	1.54	515.60	514.06	0.304	177	2740
	4	514.06	1330	1.39	410.50	409.11	0.308	248	2351
a	5	409.11	1318	1.11	391.00	389.89	0.296	266	2621
	6	389.89	1370	1.06	337.30	336.24	0.246	334	2706
	7	336.24	1651	0.91	539.60	538.69	0.326	169	2469
	8	538.69	1241	1.46	438.00	436.54	0.352	226	2539
	9	436.54	1152	1.18	422.00	420.82	0.365	244	2323
	10	420.82	1109	1.14	417.00	415.86	0.375	251	2215
	11	415.86	1080	1.13	404.00	402.87	0.373	256	2318
	Average	(last 3)	1114				0.371	250	2285

Material: Roll-Crushed Copper Slag from KHD (80 Bar)
Mill Solids Load: 1422 grams (700 cc)
Ideal Potential Product: 1422/3.5= 406.3
Ideal Circulating Load: 1422-406= 1015.7
Product Fraction in the Test Feed: 20.8%
Test Control Size: 38 μm (400 mesh)
Product 80% Passing Size: 35 μm
Feed 80% Passing Size: 470 μm

	Cycle Number	New test feed add. (g)	Number Rev.	Mass of prod. in new test feed (%)	Mass of prod. in mill disch. (%)	Net mass of prod. (g)	Prod. rate (g/rev)	Circ. Load ratio (%)	Blaine (cm2/g)
	1	1422.00	200	295.78	372.00	76.22	0.381	286	
	2	372.00	290	77.38	252.00	174.62	0.602	487	
	3	174.62	588	52.42	330.00	277.58	0.472	335	1867
	4	277.58	715	68.64	509.00	440.36	0.616	179	2192
b	5	440.36	488	105.87	422.00	316.13	0.648	247	2398
	6	316.13	491	87.78	407.00	319.22	0.650	249	1671
	7	319.22	495	84.66	402.00	317.34	0.641	254	2209
	8	317.34	503	83.62	399.00	315.38	0.626	258	2298
	9	315.38	516	82.99	424.00	341.01	0.661	246	1796
	Average	(last 3)	505				0.643	253	2101

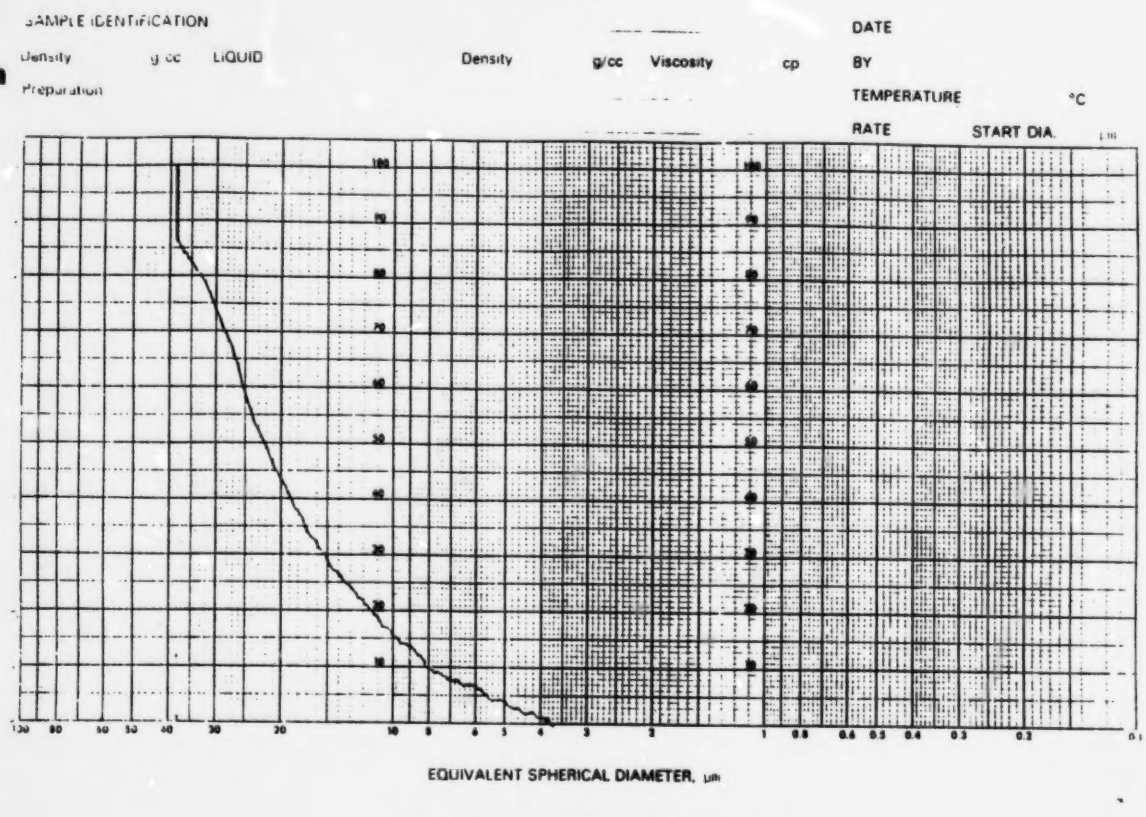
Table 8: Bond ball mill closed circuit grinding test results on the original as received (a) and the roll crushed material (80 Bar) (b).

averaged 1114 and 505 revolutions (15.9 and 7.2 minutes). Work indexes were 33.5 and 24.8 kWh/mt. Sedigraph size analysis of the Bond test products are given in Figure 2.

The results from the two Bond work index tests (closed circuit grinding) are also presented in Table 9 along with the single batch (open circuit grinding) tests carried out on the same two materials in the Bond mill apparatus. The bottom portion of this table compares the grinding times and product characteristics for one cycle of the Bond tests (at equilibrium conditions) versus the single cycle (batch) grinding. Open circuit grinding required 55 and 45.7 minutes to achieve 3000 Blaine with the original and crushed samples, respectively. The 80% product passing sizes were 43 and 38 μm using the original and roll-crushed samples, respectively. Sedigraph size analyses of the -38 μm fraction of the mill discharges are given in Figure 3.

Figure 4 shows the open cycle test results on the original and the roll-crushed samples in the Bond test mill. The 3000 Blaine was achieved at 3850 and 3200 revolutions with the original and the roll-crushed samples, respectively.

a



b

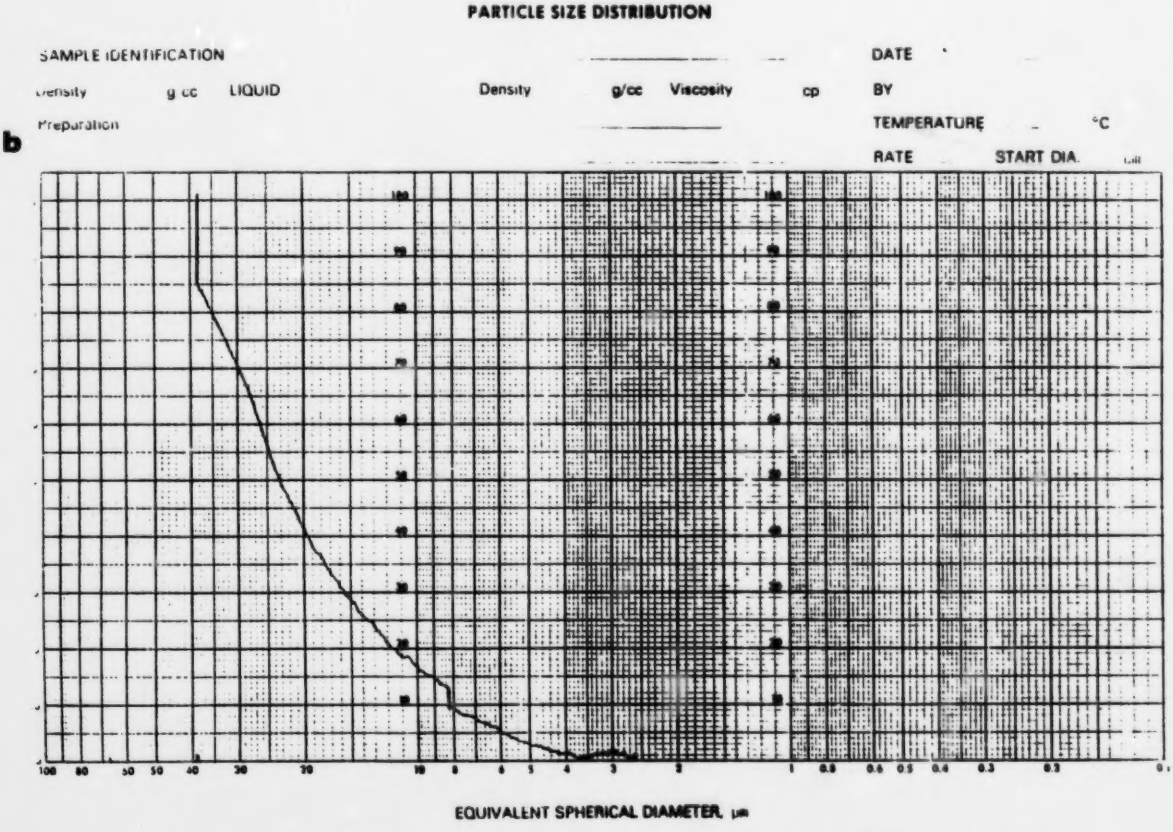
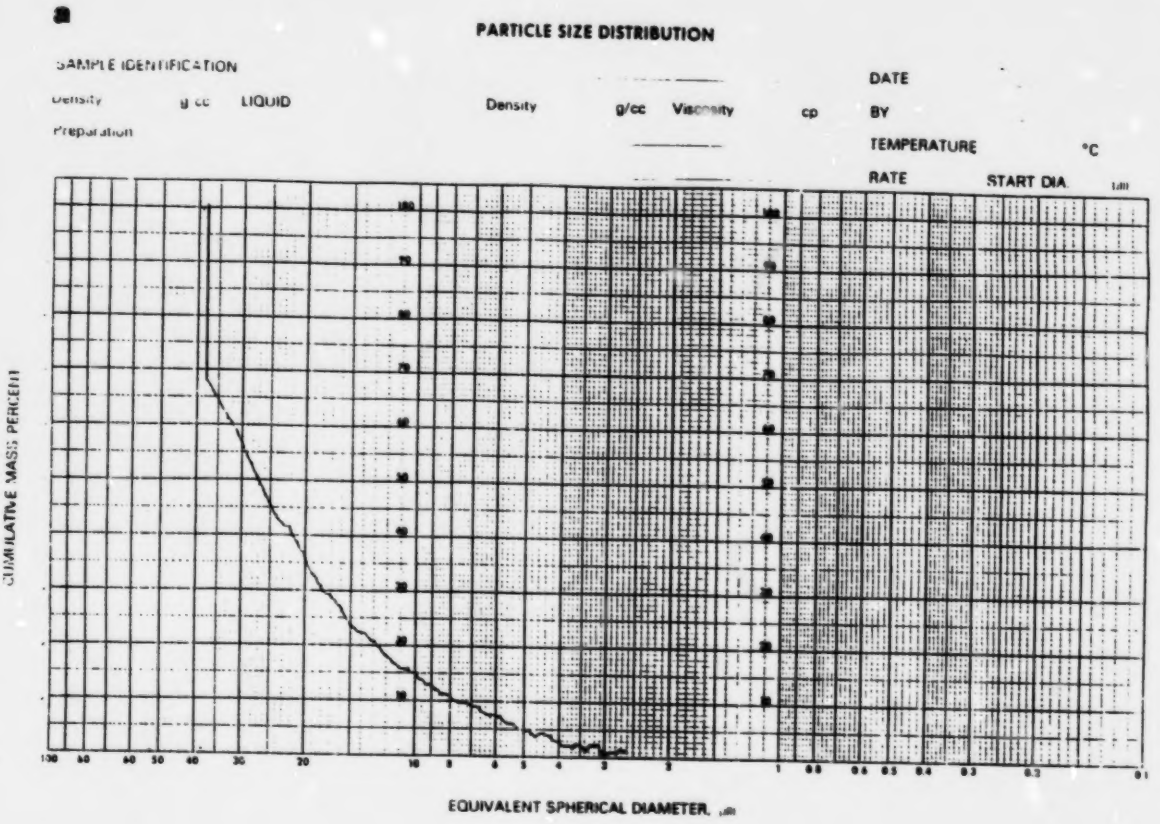


Figure 2: Product size distributions of the closed circuit grindibility tests using the original as received sample (a) and roll crushed (b) material.

Table 9: Product size distributions from open and Bond closed circuit grindibility tests performed at McGill.

Size (μm)	Cumulative % Passing			
	Closed Circuit Grinding		Open Circuit Grinding	
	Original	Roll-Crushed	Original	Roll-Crushed
2360			100.00	100.00
1700			100.00	100.00
1180			100.00	100.00
850			99.98	100.00
600			99.98	99.99
425			99.94	99.98
300			99.86	99.97
212			99.85	99.96
150			99.83	99.93
106			99.30	99.77
75			93.41	97.63
53			84.66	91.39
45			83.21	88.99
38	100.00	100.00	76.25	81.38
35	83.00	80.00	51.85	61.04
30	75.00	70.00	42.70	48.01
25	60.00	56.00	34.31	37.43
20	43.00	41.00	26.69	28.48
15	29.00	28.00	18.30	19.53
10	15.00	12.00	11.44	12.21
8	10.00	10.00	7.63	8.14
6	7.00	5.00	5.34	4.88
5	4.00	3.00	2.29	2.44
4	1.00	2.00	1.53	1.63
BLAINE (cm^2/g)	2285	2101	3000	3000
g/rev of new -38μ	0.371	0.643	0.281	0.265
GRIND. TIME (min)	15.9	7.2	55	45.7
P80 (μm)	33	35	43	38
F80 (μm)	1027	470	1027	470
WORK INDEX (kWh/mt)	33.5	24.8		
No of REVOLUTIONS	1114	505	3850	3200
MILL LOAD (g)	1422	1422	1422	1422
NEW -38 μ (g)	413	325	1080	849

a



b

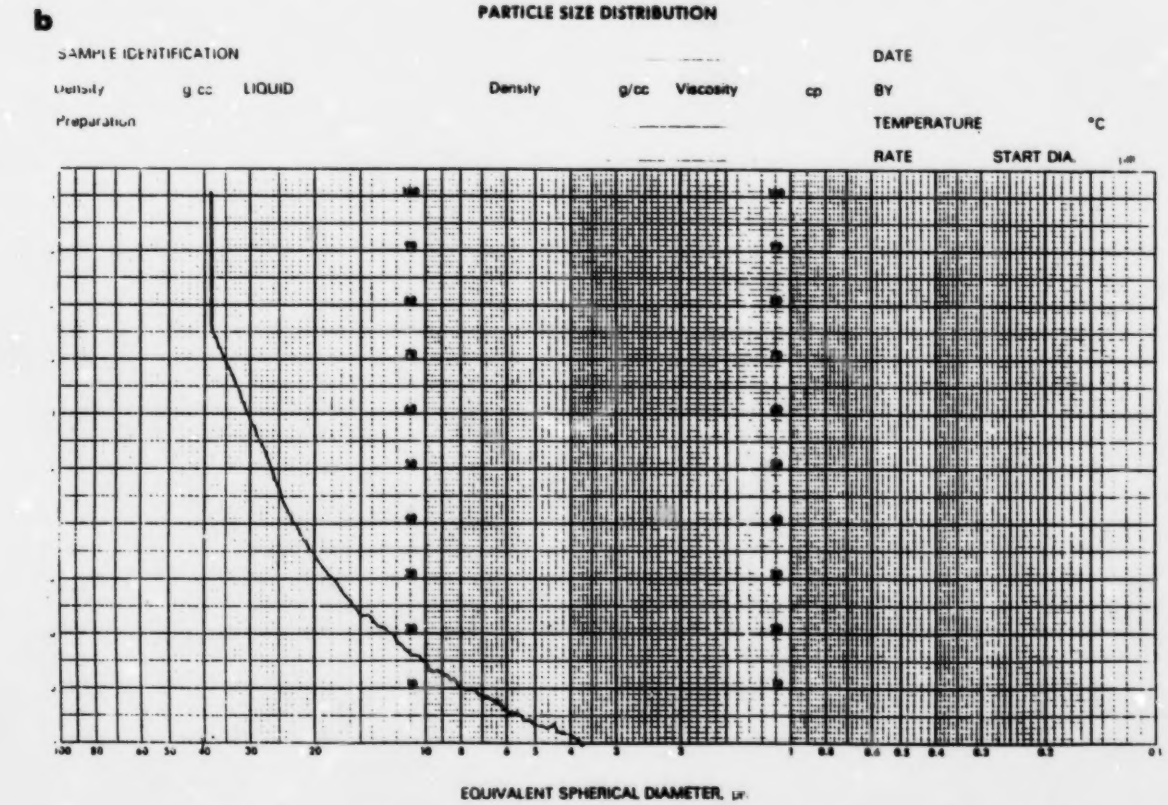


Figure 3: Product size distributions of the open circuit grindibility tests using the original as received sample (a) and roll crushed (b) material.

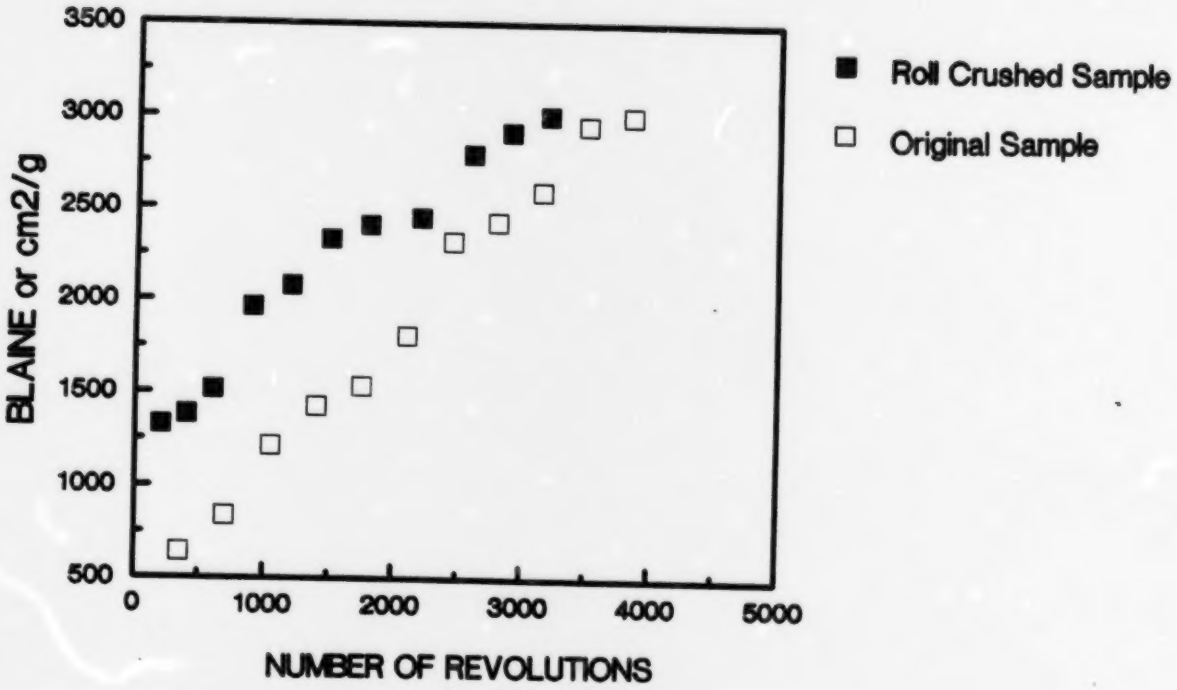


Figure 4: Open circuit grinding tests in the Bond ball mill.

PRODUCT GENERATION

The results from five-hour grinding and five-minute screening periods in the 42 cm diameter mill are given in Table 10. The batch test products were mixed in the mill for 15 minutes to make the composite sample before shipping to CANMET.

Figure 5a and 5b show the final composite product and recycle particle size distributions determined using the Sedigraph 5000D. The 80% passing size for the final composite product is 25 μm and for the recycle is 34 μm. There is 38% -10 μm in the final composite product and 28% in the recycle. Table 11 shows the size-by-size composite product and recycle finenesses. The overall final composite product and recycle finenesses were 3270 and 2581 Blaine, respectively.

Table 10: Product generation grinding results.

Test No	Grinding (min)	Screening (min)	+53 μm (g)	-53 μm (g)	Blaine (cm^2/g)
1	60	10	4052	9790	2303
	130	10	3683	11264	2674
	180	10	2856	11300	2705
	220	5	6350	7800	2767
	240	5	5868	9000	2864
	300	5	5570	8275	3108
	360	5	5300	9200	3590
2	300	5	5206	9685	3330
3	300	5	4818	9588	3113
4	300	5	5000	9389	3009
5	300	5	4950	9450	3030
6	300	5	5100	9280	3100
7	300	5	5200	9300	2980
8	300	5	5150	9370	3089

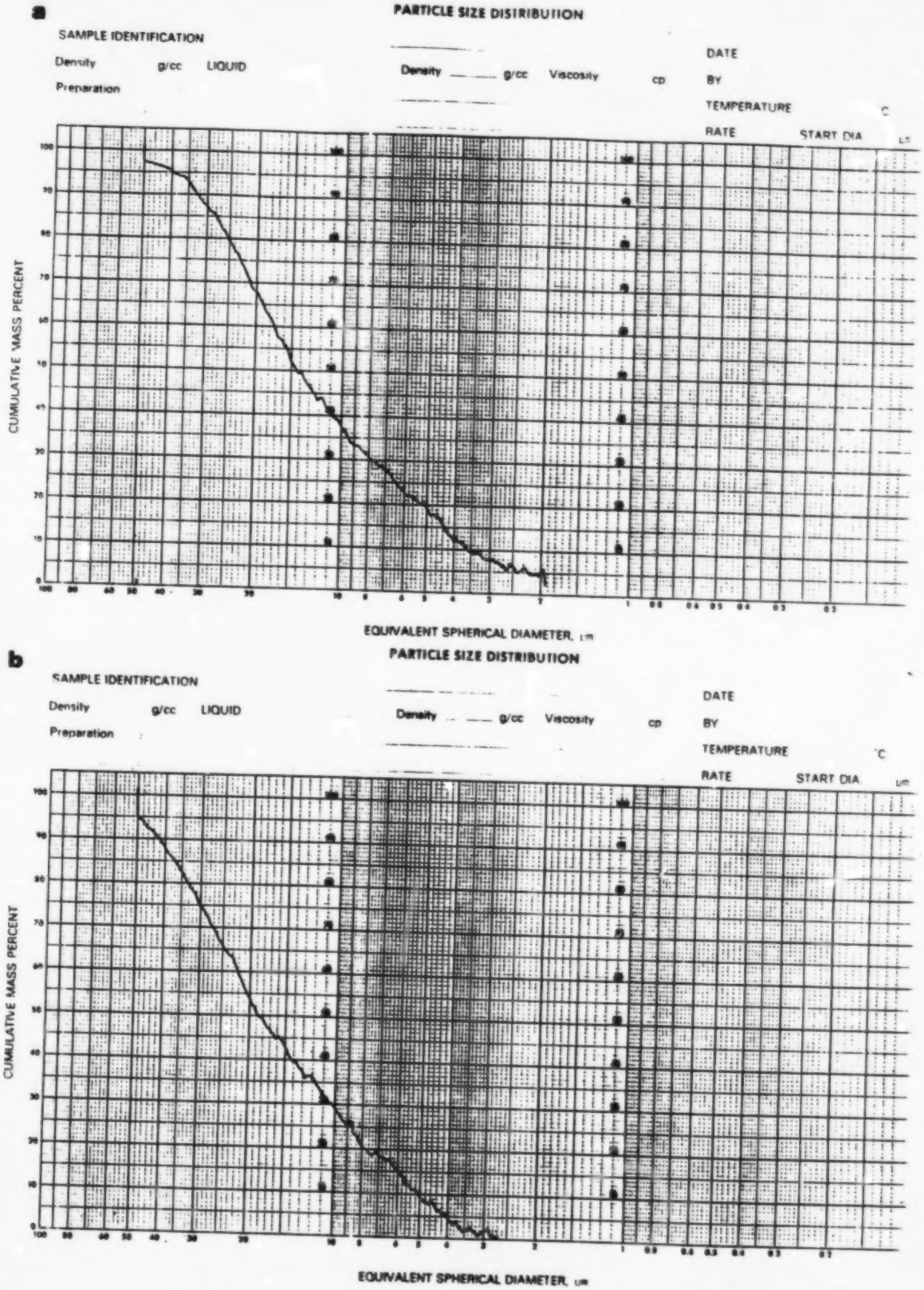


Figure 5: Final composite product (a) and recycle (b) size distributions from product generation grinding in the 42 cm mill.

Table 11: Size-by-size final composite product and oversize finenesses.

Size Fraction (μm)	Composite Product Ind. (%)	Cum. (%) Passing	Blaine (cm^2/g)	Oversize Blaine (cm^2/g)
-74 + 53	1.62	98.38	1601	1480
-53 + 45	12.76	85.62	1630	1514
-45 + 38	4.92	80.70	2600	2555
-37	80.70	-	3390	2972
Overall			3270	2581

Figure 6 shows the size-by-size product (-53 μm) fineness measurements at different grinding times for cycle 1. These measurements indicate that it is possible to achieve 3000 Blaine by selecting a 45 μm screen with 5-hour grinding or a 37 μm screen with 4-hour grinding. Overall (i.e. -74 μm) product achieves a fineness of 3000 Blaine.

The effect of screening time on product fineness can be seen in Table 10. Ten-minute screening produced a very low circulating load. Therefore, the screening time was reduced to 5 min.

INTERPRETATION OF TEST RESULTS

Industrial Mill Design Alternatives: Two industrial mill design alternatives (i.e. with and without roll crushing) can be compared for two final product finenesses: 3270 Blaine (using the P_{80} of the composite product produced at McGill, equal to 25 μm) and 3000 Blaine (using the P_{80} of KHD grindibility test, equal to 35 μm).

For 3270 Blaine, without roll crushing, the work index is calculated from the Eq.2:

$$W_i = \frac{44.5}{38^{0.23} \cdot 0.371^{0.82} \left[\frac{10}{33^{0.5}} - \frac{10}{1027^{0.5}}\right]} = 30.4 \text{ kWh/st} = 33.5 \text{ kWh/mt}$$

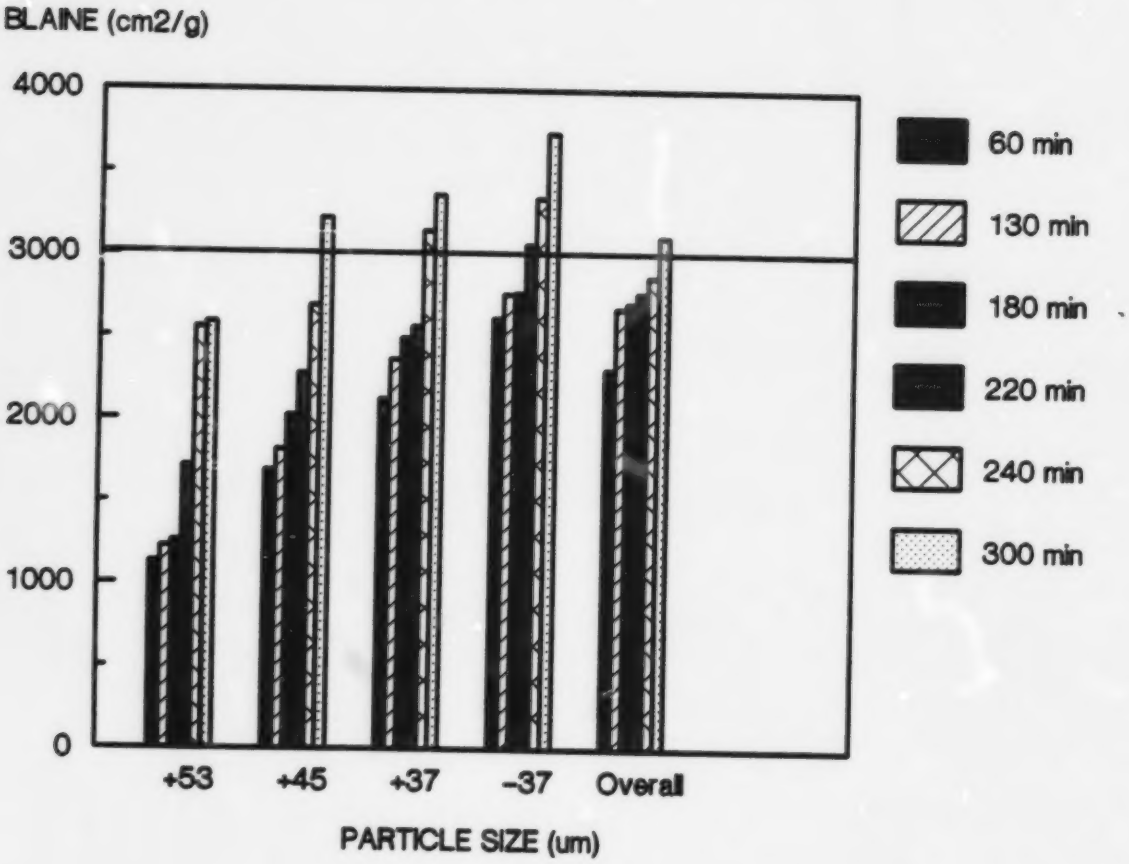


Figure 6: Size-by-size product finenesses, cycle 1, -74 μm fractions, product generation grinding in the 42 cm mill.

The specific power consumption can be calculated using the Eq.3, where P_{80} is 25 μm:

$$W = 10 \cdot 33.5 \cdot \left[\frac{1}{25^{0.5}} - \frac{1}{1027^{0.5}} \right] = 56.5 \text{ kWh/mt}$$

The full-scale grinding mill power requirements can be estimated by applying some correction factors to the calculated specific power consumption (6).

a. dry grinding is less efficient than wet grinding, a correction factor of 1.3 is applied,

b. a product fineness factor (AI) for P_{80} less than 70 μm is applied. This correction factor can be determined by the formula:

$$AI = \frac{10.3 + P_{80}}{(1.145).P_{80}} = \frac{10.3 + 25}{1.145 \cdot 25} = 1.23$$

c. a power efficiency factor is applied for change in mill diameter:

$$\text{Factor} = \left[\frac{2.44}{D_{in}} \right]^{0.2}$$

D_{in} is the mill inside diameter in meter.

This correction factor for a typical slag grinding mill (8 ft in diameter) is very close to 1.

Therefore, the commercial-scale grinding mill power without roll crushing:

$W_T = 1.3 \cdot 1.23 \cdot 1. 56.5 = 90.4$ kWh/mt

With roll crushing, the P_{80} and F_{80} are 35 and 470 μm. The work index is given by:

$$W_i = \frac{44.5}{38^{0.23} \cdot 0.643^{0.82} \left[\frac{10}{35^{0.5}} - \frac{10}{470^{0.5}}\right]} = 22.5 \text{ kWh/st} = 24.8 \text{ kWh/mt}$$

The specific power consumption can be calculated using the P_{80} of 25 μm:

$$W = 10 \cdot 24.8 \cdot \left[\frac{1}{25^{0.5}} - \frac{1}{470^{0.5}}\right] = 38.2 \text{ kWh/mt}$$

The full-scale grinding mill power is given by:

$W_T = 1.3 \cdot 1.23 \cdot 38.2 = 61.0$ kWh/mt

The full-scale grinding mill powers with and without roll crushing for the 3000 Blaine fineness can be calculated using the same work indexes and P_{80} of product obtained at KHD after grindibility tests (P_{80} = 35 μm).

Without roll crushing:

$$W_T = 1.3 \cdot 1.23 \cdot 10 \cdot 33.5 \cdot \left[\frac{1}{35^{0.5}} - \frac{1}{1027^{0.5}}\right] = 73.8 \text{ kWh/mt}$$

With roll crushing:

$$W_T = 1.3 \cdot 1.23 \cdot 10 \cdot 24.8 \cdot \left[\frac{1}{35^{0.5}} - \frac{1}{470^{0.5}}\right] = 48.7 \text{ kWh/mt}$$

Table 12a summarizes the predicted industrial full-size grinding equipment energy consumptions to achieve a product finenesses of 3270 and 3000 Blaine using the Bond methodology. Comparative results using the Zeisel methodology are given in Table 12b

Table 12: Comparison of predicted full-scale power consumptions, Bond methodology (a) and Zeisel methodology (b).

a Energy Consumption (kWh/mt)	Without Roll Crushing	With Roll Crushing	Blaine (cm^2/g)
Roll Crushing	-	5.5	
Ball Milling	90.4	61.0	
TOTAL	**90.4**	**66.5**	**3270**
Roll Crushing	-	5.5	
Ball Milling	73.8	48.7	
TOTAL	**73.8**	**53.2**	**3000**
b Energy Consumption (kWh/mt)	Without Roll Crushing	With Roll Crushing	Blaine (cm^2/g)
Roll Crushing	-	5.5	
Ball Milling	63.0	48.0	
TOTAL	**63.0**	**53.5**	**3000**

Closed versus Open Circuit Ball Milling: Direct comparison of the closed versus open grinding test results in the Bond apparatus (Table 2) is not possible because of the differences in the Blaine fineness of the final test products. However, because the material was tested for Blaine fineness throughout the grinding periods during the open circuit tests (Figure 4), it is possible to compare the test results where the Blaine finenesses were the same as the closed Bond circuit products, i.e. 2285 Blaine for the original sample, and 2101 Blaine for the roll crushed sample. The results are shown in Table 13 and 14.

Table 13: Closed versus Open Circuit Grinding to 2285 Blaine in the Bond Apparatus, Original Sample.

	Closed Circuit	Open Circuit
Blaine (cm^2/g)	2285	2285
Grinding Time (min)	15.9	37.9
Product Mass (g)	414	1422
g/min of 2285 Blaine Material	**26.0**	**37.5**

Table 14: Closed versus Open Circuit Grinding to 2101 Blaine in the Bond Apparatus, Roll Crushed Sample.

	Closed Circuit	Open Circuit
Blaine (cm^2/g)	2101	2101
Grinding Time (min)	7.2	17.1
Product Mass (g)	408	1422
g/min of 2101 Blaine Material	56.7	83.2

In both cases, the production rate of material of similar Blaine fineness is much higher in open circuit (single batch) grinding.

Another method to evaluate grinding efficiency is the production rate of new -38 μm material in closed versus open circuit grinding. For the closed circuit tests, this is 26.0 (original sample) and 45.1 (roll crushed sample) grams per minute, respectively.

For the open circuit tests, it is necessary to estimate the amount of -38 μm material produced at the respective grinding times of 37.9 and 17.1 minutes. This can be done by calculating a disappearance rate constant, k, in the kinetic equation.

$$R = R_0 \cdot e^{-kt}$$

where R is the amount of oversize (+ 38μm) remaining at time t.

R_0 is the amount of oversize at t=0.

This first order equation is normally applied to relatively short period grinding tests (8). It is used here to provide approximate grinding rates for comparison of these test results only.

For the original sample ground in open circuit:

R_0 = 1422g . (100 - 0.27)% = 1418g

At t = 55 minutes, R = 1422g . (100 - 76.25)% = 338g
Therefore:
$338 = 1418 . e^{-55k}$
Solving, k = 0.02607

At t = 37.9 minutes:

$R = 1418 . e^{-37.9(0.02607)} = 528g$

The amount of -38 μm produced is therefore:

1422 - 528 - 1422(0.27)% = 890g

The grams per minute of 38 μm material produced is therefore:

890/37.9 = 23.5 grams per minute.

Similarly, for the roll crushed sample ground in open circuit:

R_0 = 1422 . (100 - 21.67)% = 1114g

At t = 45.7 min, R = 1422 . (100 - 81.38)% = 265g
In this case, k = 0.23788
At time t = 17.1 minutes, R = 651g
The amount of -38 μm produced is therefore:
463/17.1 = 27.1 grams per minute

The results are summarized in Table 15.

Table 15: Closed versus Open Circuit Production Rates of New -38 μm Material in the Bond Apparatus.

	Closed Circuit	Open Circuit
Original Sample		
Blaine of Product	2285	2285
Grinding Time (min)	15.9	37.9
Production Rate of New -38 μm Material (g/min)	**26.0**	**23.5**
Roll Crushed Sample		
Blaine of Product	2101	2101
Grinding Time (min)	7.2	17.1
Production Rate of New -38 μm Material (g/min)	**45.1**	**27.1**

These results show that in terms of the production rate of new -38 μm material, of the two types of circuits producing the same Blaine fineness in the product, the closed circuit is more efficient than the open circuit, particularly for the roll crushed sample.

CONCLUSIONS

1. Assuming 5.5 kWh/mt is consumed in roll crushing, Zeisel grindibility tests at KHD indicate a reduction in total energy consumption from 63 kWh/mt for ball milling alone to 53.5 kWh/mt for roll crushing plus ball milling (a savings of 15%) to produce 3000 Blaine product fineness on a commercial scale. On the same bases, the Bond work index tests indicate a reduction in total energy consumption from 73.8 kWh/mt for ball milling alone to 53.2 kWh/mt for roll crushing plus ball milling (a savings of 28%). The difference in the relative percentage energy savings is due to the different energy requirment estimated by the two scale-up methods for ball milling alone. Due to the unusual size distribution characteristics of the granulated material (compared to a stage-crushed or ground product commonly used in Bond work index testing), it is difficult to say which is more accurate. However, the two methods yield virtually identical results for the roll crushing plus ball milling flowsheet.

2. Bond ball mill work index tests showed that the work index of the slag fell from 33.5 to 24.8 kWh/mt (a reduction of 26%) as a result of roll crushing.

3. In terms of the production rate of material to a specified Blaine fineness, open circuit (single batch) grinding in the Bond apparatus was significantly more efficient than closed circuit (locked cycle) grinding.

4. In terms of the production rate of new -38 μm material, closed circuit grinding in the Bond apparatus was more efficient than open circuit (single batch) grinding.

REFERENCES

1. Douglas, E., Malhotra, V.M., and Emery, J.J., "Cementitious Properties of Nonferrous Slags from Canadian Sources", Cement, Concrete and Aggregates, ASTM, Summer 1985.

2. Cohen, S.M., and Porter, E.S., "Hydraulic Roll Crusher", AIME Meeting, St. Louis, Sept. 1986.

3. McIvor, R.E., "Classification Effects in Wet Ball Milling Circuits", Mining Eng., Aug. 1988.

4. Kirkpatrick, E., "Production of a New Granulated Blast Furnace Slag at Reiss Lime Company of Canada Ltd.", Inter. Workshop on Granulated Blast Furnace Slag in Concrete, CANMET, Toronto, Oct. 1987.

5. Bond, F.C., "Crushing and Grinding Calculations", Brit. Chem. Eng., Jun. and Aug., 1961.

6. Rowland, C.A., "Selection of Rod Mills, Ball Mills, Pebble Mills and Regrind Mills" in the "Design and Installation of Comminution Circuits", Eds. by Mular, A.L., and Jergensen, G.V., SME-AIME, 1982, pp. 393-438.

7. Atkinson, R.J., Hannaford, A.L., Harris, L., and Philip, T.P., "Using Smelter Slag in Mine Backfill", Mining Magazine, Aug. 1989, pp. 118-123.

8. Magdalinovic, N., "A Procedure for Rapid Determination of the Bond Work Index", Inter. Journal of Min. Proc., Vol. 27, 1989, pp. 125-132.

APPENDICES

KHD HUMBOLDT WEDAG

Laboratory Report

about

roller press tests of copper slag from CanMet for the department of Mining and Metallurgical Engineering of McGill University, Montreal, Quebec, Canada

Project No.: 9 8444 9 0001

Laboratory No. :	ZL 6490
Receipt of samples :	June 23, 1989
Number of samples :	1
Designation of samples :	copper slag
Report written by :	Dr. H. U. Schäfer
Department :	IT - AZ 5
Date :	31.07.89

All tests are carried out in our laboratories with utmost care, however without any liability on our side.

Humboldt Wedag Division of KHD Canada received a purchase order for roller press tests of copper slag of CanMet through the department of Mining and Metallurgical Engineering of McGill University, Montreal, Quebec, Canada.

The sample of the copper slag arrived in the Research and Development Center of KHD Humboldt Wedag AG on June 23, 1989.

The copper slag had been packed in three 5 gallon barrels. The total weight was approximately 100 kg.

The sample had to be passed through the roller press and the grindability of the original samples and the desagglomerated flakes had to be performed. The grindability test had to be carried out on a mass related surface of 3000 cm^2/g according to Blaine.

A specimen of the desagglomerated flakes had to be sent back to McGill University.

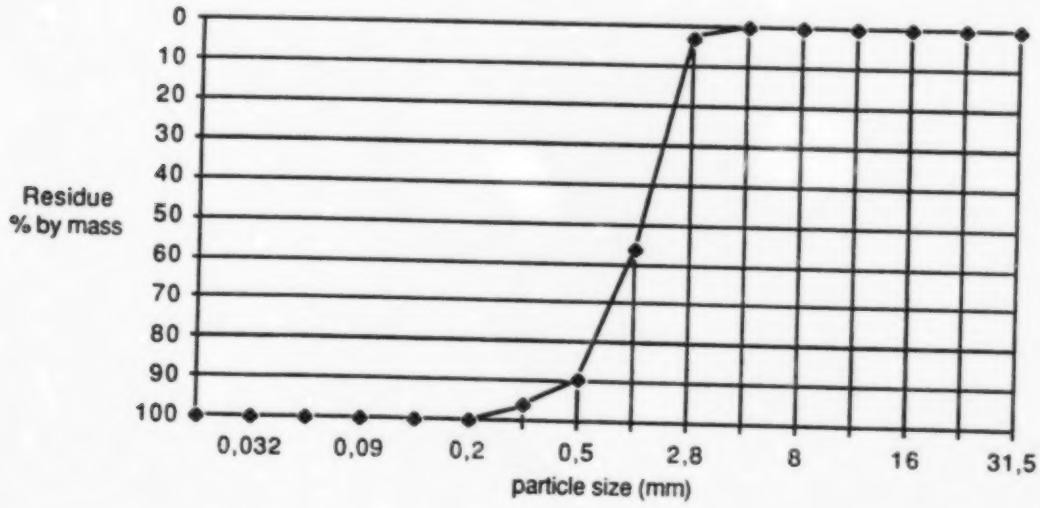
The test procedures and the results are described hereafter.

1. Description of the sample

The copper slag was a dark coloured material.

It had been delivered in a very narrow particle size range between 5,6 mm and 0,2 mm. The exact particle size distribution is listed in the tables attached to this report.

Rollerpress test of copper slag



The particles of the granular slag were mainly regular shaped , partly rounded, and edged to sharpedged.

The moisture content upon delivery was 2,6 % by mass. The roller press tests have been carried out without drying of the sample.

2. Roller press test procedure

The sample has been splitted into several parts.

With one part the properties of the original sample have been determined: the grain size distribution has been determined and a grindability test has been performed.

The other part of the sample is used for roller press tests. In order to study the influence of the pressure in the gap of the roller press on the comminution efficiency 2 tests with 2 different standard forces f (force per cm of roller width and 1 m roller diameter) have been carried out. The tests are carried out with a laboratory roller press type RP 80-12.

The flakes obtained from each test are desagglomerated in a hammer mill with a grate of 9 mm x 32 mm.

The grain size distribution of the desagglomerated flakes is determined and a grindability test is performed.

3. Roller press tests

3.1 Test 1

The first test has been carried out after adjusting of the accumulators of the roller press to a pressure of 80 bar.

36 kg of copper slag (one 5 gallon container) had been passed through the press.

The nipping characterisitics of the copper slag were moderately good.

The standard pressure recorded was 94 kN / cm roller width and 1 m roller diameter.

The thickness of the flakes was 7 mm to 8 mm and the width of the flakes was around 90 mm. However we observed a considerable amount of unpressed materials from the edge zone of the rollers. Due to this the percentage of flakes retained was 38 % by mass of the feed material only. This has to be considered as a matter of the test roller press itself, whose roller width is only

120 mm. Larger roller width used in industrial sized roller presses will produce a higher percentage of flakes most probably. Furthermore the surface of the rollers of the test roller presse were smooth, while industrial sized machines can be equipped with Chevrons on the roller surfaces for improved nipping of the feed.

The density of the flakes was 3,11 g /cm^3.

3.2 Test 2

The second test has been carried out after adjusting of the accumulators of the roller press to a pressure of 60 bar.

30 kg of copper slag (one 5 gallon container) had been passed through the press.

The nipping characterisitics of the copper slag were not as good as in the first test.

The standard pressure recorded was 71 kN / cm roller width and 1 m roller diameter.

The thickness of the flakes was 7 mm to 9 mm and the width of the flakes was around 90 mm. Only 30 % by mass of the feed had been retained as proper flakes. Their density was slightly lower than in the first test and reached 3,06 g/cm^3.

For the evaluation of both tests only the flakes have been used. The unpressed material from the edgezones has been separated from the roller press product.

4. Evaluation of test results

4.1 Grindability test

The grindability of the original sample and the desagglomerated flakes from both tests had been determined according to the open-circuit-principle with a testing equipment of Messrs. Tonindustrie, Berlin (Zeiseltest).

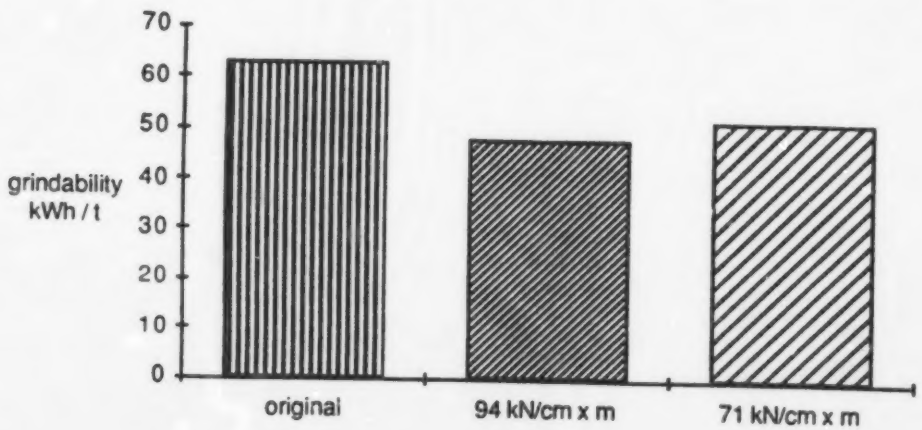
About 60 g of each sample in an particle size range between 0,75 mm and 1,0 mm has been prepared. This sample is fed into the grindability tester and ground in 7 to 8 steps until a mass related surface of more than 3000 cm^2/g according to Blaine has been recorded.

At the requested mass related surface of 3000 cm^2/g the following readings have been evaluated :

	specific energy consumption kWh/t	saving kWh/t
original sample	63	
sample from test 1	48	- 15
sample from test 2	52	- 11

These values are applicable for dry grinding method.

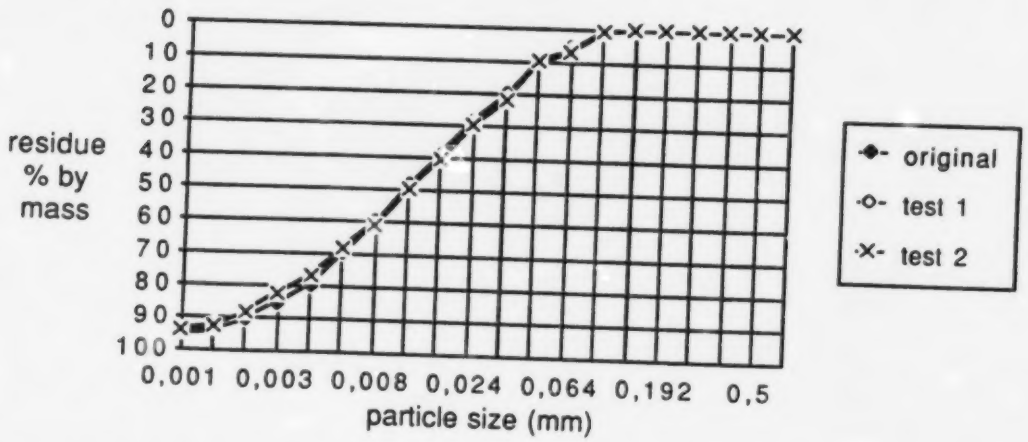
grindability of original and flakes



It is obvious, that with higher forces of pressing the specific energy consumption for grinding of the flakes is reduced.

The particle size distribution of the samples after the the grindability test has been determined by means of a Cilas Lasergranulometer .

Particle size distribution after grindability test



4.2 Particle size distribution of flakes

It is not only the reduced specific energy consumption but also the particle size distribution of the desagglomerated flakes which contribute to the high economic effort of the roller press.

The particle size distribution of the desaggloreated flakes show, that by the roller press a drastic increase of fines is achieved. On 0,2 mm aperture size 47 % by mass residue were recorded, and on 0,090 mm aperture size the residue was approximately 66 % by mass.

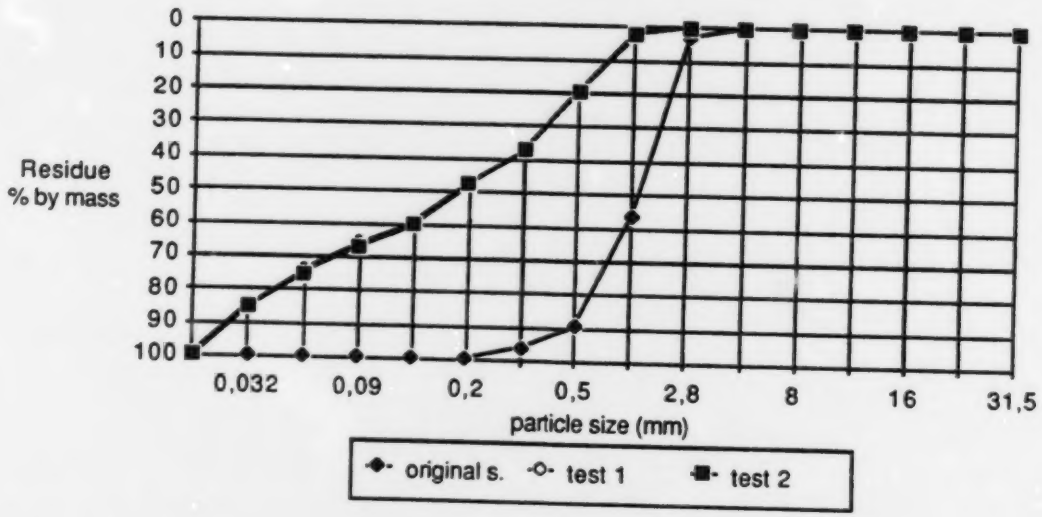
Contrary to the specific grindability no influence of the pressing force on the the particle size dirstibution has been observed.

KHD HUMBOLDT WEDAG

Laboratory no. : ZL 6490
Project : CanMet , Copper slag
Project No .: 9 844 4 9 0001

Date : 31.07.89

Rollerpress test of copper slag



5. Return of samples

A mixture of desagglomerated flakes in the ratio 1 by 1 of test one and test two has been airfreighted to McGill University, Montreal, on July 13, 1989 under air way bill no. 018 2879 8276 through M/S Huppertz Air Cargo Service .

IT - AZ 5

Dr. H. U. Schäfer

KHD HUMBOLDT WEDAG

Laboratory no. : ZL 6490
Project : CanMet , Copper slag
Project No .: 9 8444 9 0001

Date : 31.07.89

Rollerpress test of copper slag

grain size mm	Residue % original s.		Residue % test 1	Residue % test 2
31,5				
22,4				
16,0				
11,2				
8,0				
5,6				
2,8	2,9			
1,0	56,3		1,9	2,4
0,5	89,4		19,6	19,6
0,315	96,2		37,2	37,2
0,200	100,0		47,0	47,6
0,125	100,0		59,5	60,1
0,090	100,0		65,9	66,8
0,063	100,0		74,2	75,0
0,032	100,0		84,8	85,2
0,000	100,0		100,0	100,0

KHD HUMBOLDT WEDAG

Laboratory no. : ZL 6490
Project : CanMet , Copper slag
Project No .: 9 8444 9 0001

Date : 31.07.89

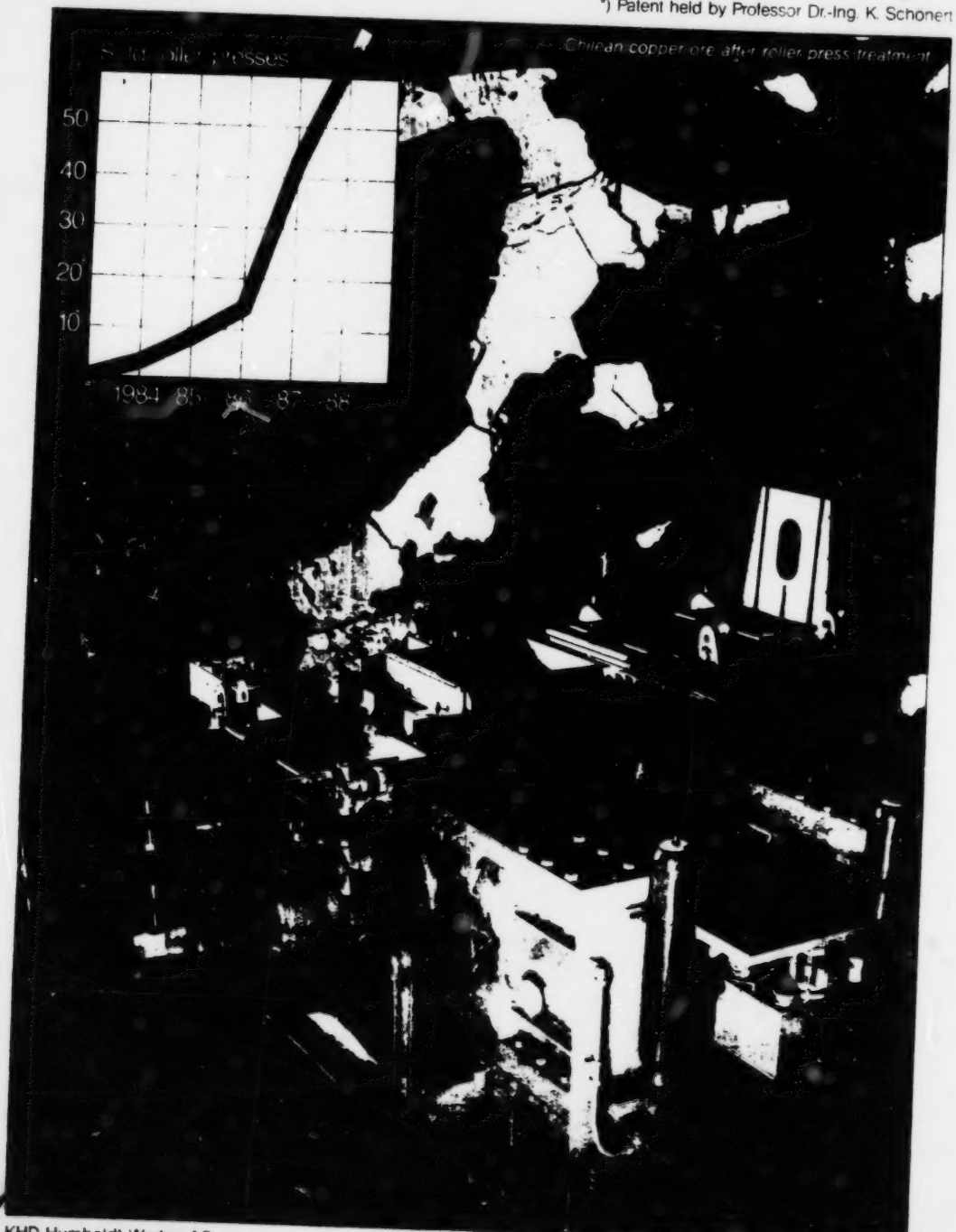
Grain size distribution : copper slag after grindability test

grain size mm	% by mass original	% by mass test 1	% by mass test 2
0,0010	95,0	94,1	94,3
0,0015	93,8	92,4	92,6
0,002	90,6	88,2	88,4
0,003	85,6	82,3	82,5
0,004	79,7	76,5	76,8
0,006	70,3	68,1	68,8
0,008	61,0	60,2	61,1
0,012	48,6	48,6	49,8
0,016	39,1	40,0	41,0
0,024	28,5	28,9	29,8
0,032	20,3	21,3	22,0
0,048	10,1	9,7	10,2
0,064	5,9	6,6	6,9
0,096	0,9	0,0	0,5
0,128	0,0	0,0	0,0
0,192	0,0	0,0	0,0
0,250	0,0	0,0	0,0
0,315	0,0	0,0	0,0
0,500	0,0	0,0	0,0
1,000	0,0	0,0	0,0

The Humboldt Wedag roller press – a great success!

Worldwide, plant operators prove daily our roller press technology*) is working for improved ore liberation as well as substantial energy saving in comminuting cement clinker and many other minerals. Our experience gained from hundreds of varied materials and sales exceed 50 roller presses continue to provide a solid basis for operational and process reliability for your economic efficiency and success.

*) Patent held by Professor Dr.-Ing. K. Schonert



Chilean copper ore after roller press treatment

KHD Humboldt Wedag AG
Postfach 910457, D-5000 Köln 91
Tel. (0221) 823-0, Telex 8812-0, Telefax (0221) 823-7240
Federal Republic of Germany

HUMBOLDT WEDAG

KHD
HUMBOLDT
WEDAG

c.c. -> M. Kaya
E. Dengas

TELEFAX

to : KHD Canada, attn. Geoff Ward
Telefax No. : (001514) 33 24 173
Page : 1 of 3

from: KHD Humboldt Wedag AG , D-5000 Cologne 91, Fed.Republic of Germany
Dr. Schäfer , Dept. IT - AZ 5 , Rawmaterialtechnology
Phone : (0221) 8 23 67 05 , **Telefax : (0221) 8 23 66 27**
Telex : 88 12 261

Ref.: CanMet Copper Slag
Date : September 1, 1989

Thanks for your telecopy regarding the a.m. project. Please transmit the following information to Mr. McIvor of GDP.

Para 1 : The result of the Zeisel test gives the specific energy consumption of the ball mill only realted to the power required on the motor coupling of the mill tube itself.
What the power consumption of the roller press is concerned we suggest to calculate with 5 kWh/t (metric) to 6 kWh/t. It's difficult to give a more exact figure, because we have tested samples like that only few times. From our experience with blast furnace slag we have derived the figures given above.

Para 2 : In a full scale roller press you can expect a recovery rate for flakes in the range between 90 % and 95 %.

Para 3 : standard pressure means the force in the gap as per 1 cm roller width and 1 m roller diameter.

P.2 3
6754:#
3324173
REC. FAR: TELECOPIER AERO. 7011 ; 9- 1-89 12:02PM ;
SEP 01 '89 11:59 KHD CHHEH 3324173

TELEFAX

from : KHD Humboldt Wedag Ag, D-5000 Cologne 91, Fed.Republic of Germany
Dr. Schäfer , Dept.: IT-AZ 5 , Rawmaterialtechnology
Phone: (0221) 823 67 05 , Telefax : (0221) 823 66 27, Telex : 88 12 261

When you look on the design of the roller press, you will realize the accumulators on one side of the machine. Those accumulators are adjusted to a certain pressure, in our example 80 bar. These accumulators have to keep the pressure in the hydraulic system on a certain level. When we feed the press with material the loose roller moves in order to react on the forve of the feed in the gap. The hydraulic system takes up the pressure and keeps the roll in an almost constant position. This pressure is recorded and converted to the "standard force" , which means force per 1 cm of gap width and 1 m roller diameter.

Para 4 : Test 1 has been carried out with the accumulators set to 80 bar. On the higher pressure you get an easier to grind product.

52 kWh/t at 60 bar is correct.

Meanwhile we have received the second sample from you. We will pass it through the roller press sometime next week and then we will send it back immediately without carrying further tests on it.

Kind regards

Dr. H. U. Schäfer

P.3/3
SEP 01 '89 12:00 KHD CHEMIE 3324173
REC.PAR:TELECOPIER XEROX 7011 ; 9- 1-89 12:04PM ; 3324173-
6754:# 3

Grindability test for cement raw materials according to Zeisel (closed circuit grinding combined with air separator)

Meant for ascertaining the energy required for comminuting limestones, calcareous marl, marl, etc.

Set-up of the grindability tester (type 'Tonindustrie')

The material to be tested is subjected with the aid of grinding media primarily to rolling and frictional forces in a narrowed grinding pan. This is a comminuting method which is of particular importance for fine grinding. The grinding balls do not move unhindered but are guided by a grinding ring which simultaneously transmits the necessary grinding pressure. In order not to complicate or even preclude calculation of the expenditure of energy, a slip - if any - between balls and grinding ring is neglected. Measuring is done via the supported grinding pan which has been connected to a spring-type dynamometer and an electric measuring device.

Description of the equipment

The equipment comprises a heavy cast housing (a) (fig. 1) which accommodates the drive, the gearbox and the measuring device. The friction gear transmits the torque of the three-phase motor (m) (220/380 volts, 50 Hz, 0.8 kW) onto a plunger to which the grinding ring (b) has been mounted. The plunger with grinding ring can be lifted and lowered with the aid of lever (c). When in operating position, the grinding ring rests with a load of 25 kgs on eight steel grinding balls (k) (25 mm dia.) which balls run along a circular grinding path in the grinding pan (d). The load of 25 kgs can be increased by fitting additional weights (e). The grinding pan is carried in a rotatable manner and retained in zero position by a spring-type dynamometer which latter is positioned around the axle of the grinding pan. The measuring device has been incorporated in the instrument base and consists of potentiometer, voltage stabilizer with rectifier and ampere-hour meter (h).

Fig. 1

Grindability tester - schematic representation

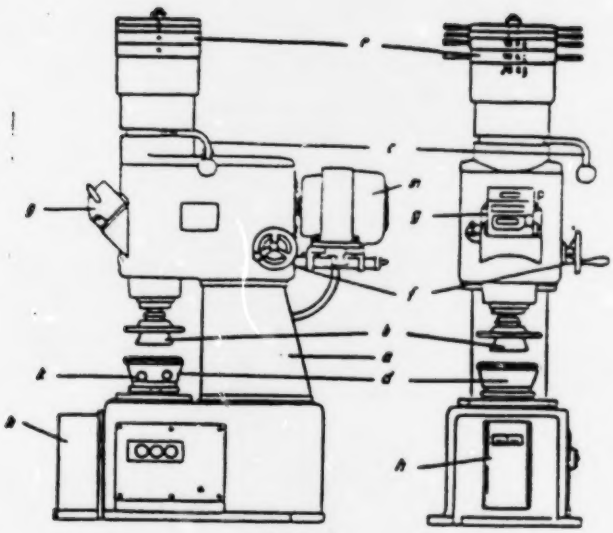


Bild 1
Mahlbarkeitsprüfer, schematische Darstellung

Test procedure

The following conditions will definitely have to be adhered to:
The feed quantity for the first grinding operation has to equal 25.0 g (about 50 - 60 g shall be prepared). The feed size shall be 0.75 - 1.0 mm. A speed of n = 200 rpm shall be set (duration of each grinding operation 0.5 min.)
The pressure load amounts to 25 kgs.

The tests are carried out by an imitated closed-circuit grinding operation. The feed size is 1.0 to 0.75 mm. 25 g of material of that size are fed into the grinding pan and ground at intervals of 30 seconds (= 100 revolutions). The revolutions are read at the counter; the complete sample material is then removed from the grinding pan and separated into finished product and rejects at a cut point of 0.125 mm. The rejects are replenished with prepared feed material by the quantity of finished product removed by screening so as to obtain again 25 g of material; that quantity is again filled into the grinding pan.
The size of the finished product is ascertained by the residues remaining on a screen of 0.09 mm mesh width (4,900 M/m^2). That operation is repeated until the finished product equals about 25 g - corresponding to the feed quantity. The specific working index determined for the single fractions is combined and related to exactly 25 g.

Results of measuring

The data ascertained are entered in the corresponding forms.

sample no. identific.	grind. time min.	counter units	cumul. Nm	cumul. Nm	screen fin. product 0.125mm g	throughs cumul. qty. 0.125 mm g	qty.resi of fin. product 0.09 mm g
	0.5						
	1.0						
	1.5						
	2.0						
	2.5						
	average					25.000	
				kWh/t at			% R 0.09 mm

Test procedure

1. The sample shall be mixed and a representative average sample taken (quartered).

2. Sample preparation

   - the average sample is heedfully reduced (e.g. in a jaw crusher, disk mill, or the like)
   - the sample is screened after each comminution operation; screen size 0.75 - 1.0 mm
   - the oversize material is further reduced; sample quantity to be prepared - approx. 60 g

3. Test procedure

   - weighing of the material quantity of 25.0 g
   - filling that quantity into the grinding pan
   - lowering the weight
   - carrying out grinding
   - removing the sample
   - screening the complete sample quantity at 0.09 mm (ALPINE air jet screen)
   - recording quantity of residues in g
   - screening the residues at 0.125 mm (ALPINE air jet screen)
   - recording quantity of residues in g
   - replenishing the residues retained on 0.125 mm with fresh material of 0.75 - 1.0 mm size until 25 g are obtained and repeating the grinding test
   - grinding as described above is repeated until the quantity of finished product (cumulative quantity of throughs - 0.125 mm) equals approx. 25 g, i.e. until it is identical to the feed quantity.

4. Evaluation

   The specific working index determined for the single grinding operations is combined and related to 25 g.

5. Remark

   The results of grindability tests will be informative and reliable only if all preconditions specified have been strictly adhered to.

Grindability testing and tube mill design for cement raw materials

1. Importance of grindability testing

Reliable testing methods and theoretical fundamentals are a prerequisite for the mill construction engineer for exactly dimensioning his systems so that he will no longer have to take as a basis merely empirical values.

It is generally known that grindability testing 1-4) can provide the prerequisite basis.

The tube mill manufacturer is permanently confronted with the task of building a mill so that - at a competitive price - it will meet certain, often strict guarantee requirements with regard to throughput, energy consumption and fineness of the final product.

As the grinding capacities/individual system have continuously increased during recent years, the financial risk of an incorrect design has similarly increased.

The figures given below illustrate the total economic significance of a correct tube mill design.

In 1973 the cement production rate worldwide equalled approx. 690 million tons, i.e. apart from cement clinker including 65 million t/a of slag cement, approx. 1.1 billion tons of raw material had to be ground. The increase in cement production rate/year during recent years was such 5) that it can be expected that at present approx. 40 million t/a of cement and 64 million t/a of raw material will have to be produced in addition per year than for every year previously. For this additional amount a new mill capacity will have to be provided for every year. Moreover, the mill capacity which must replace old, worn-down mills, must be taken into consideration.

Grindability testing of cement r a w m a t e r i a l s is of particular problematic nature, since these are often not so hard, brittle and macroscopically homogenous as properly burnt cement clinker.

More than a dozen of grindability testing methods are known, of which 3 appear to be specially suitable for grinding cement raw materials. These are the Hardgrove 6), Zeisel 1-2) and Bond 7-9) testing methods. These and other methods have been described in detail by Remenyi 10) and Beke 11). A test of the grindability of the cement raw materials of various geologic and geographic origin is intended for each of these three methods for which the results are checked for suitability for dimensioning of tube mills.

2. Material properties and preparation of samples of the cement raw materials

The samples to be tested are limestone, lime marl, marl in the real sense and loess from quarries of German cement plants (Table 1). All samples were at first dried in drying ovens at 100 oC, for which the moisture of the material as delivered and later the residual moisture were determined at the same time. For ascertaining the grain composition, test sieving in accordance with DIN 4188 was done on a vibratory screening machine. The grain composition of the different samples as delivered is shown in Fig. 1. This size distribution of the material as delivered is the feed size for industrial-scale mills in the respective cement plants. All samples were comminuted carefully to a particle size less than 3.3 mm and afterwards again subjected to a screen analysis. See Table 2 below.

TABLE 2
Chemical composition of cement raw materials

Sample	Loss on ignition	SiO_2 (%)	Fe_2O_3 (%)	Al_2O_3 (%)	$CaCO_3$ (%)	$MgCO_3$ (%)	CaO (%)	MgO (%)
A	33.96	15.85	1.54	4.48	73.0	2.33	40.94	1.11
B	41.73	2.20	0.50	1.24	92.5	2.52	51.87	1.25
C	35.47	14.58	0.96	3.06	79.6	0.63	44.51	0.30
D	21.81	39.55	2.44	6.35	44.8	3.82	25.10	1.82
E	36.57	10.32	1.54	3.60	75.0	6.77	42.06	3.23
F	35.60	12.26	1.96	4.36	78.0	1.70	43.74	0.81
G	35.49	12.12	1.96	4.58	77.6	1.81	43.47	0.86
H	36.10	11.34	2.66	4.30	79.0	0.84	-	-
I	39.81	6.41	0.92	2.64	86.4	2.48	-	-

Subsequently determined for all samples were the specific gravity by means of pycnometer, the scratch hardness as per Mohs, the bulk density - loose and vibrated - and the chemical composition. All values have been included in Tables 1 and 2.

The samples for grindability testing as per Hardgrove and Zeisel were manually comminuted in a mortar to the respectively necessary particle size. A ground product of fraction 1.19 - 0.59 mm was produced for a double Hardgrove test; the feed material for the grindability tester as per Zeisel had to be delivered in fractions from 1.0 mm to 0.75 mm. For crushing in the mortar a passage of the overall mortar feed through the 1.19 mm and/or the 1.0 mm screen had to be made sure to avoid selective crushing in the mortar.

3. Grindability testing of cement raw materials and results obtained

3.1 Hardgrove method 6)

The Hardgrove method has been developed in the USA for determination of the grindability of pit coal. For this duty it has been applied as standard method for a long time. Based on the coal of the Jerome mine, upper seam, Kittanning, county Somerset, Pennsylvania with a grindability index of 100oH (= Hardgrove) it serves as a comparison.

Fig. 3 shows the grindability tester as per Hardgrove. A cast steel bowl contains 8 steel balls b of 1" (25.4 mm) diameter which are rotated by grinding ring c arranged above. Via shaft d and a pair of gears e and f located outside the housing, the grinding ring is driven by a geared motor g of 0.2 kW. The grinding shaft has a capacity of 19 rpm. Weight has been slipped on the grinding shaft. It has been dimensioned such that the total weight of shaft d, gear e, rings i and m, grinding ring c and loading weight amounts to 29 kp. Rings i and m are guide rings to which release lever p and q linked to counter o have been fastened. As soon as the number of overall revolutions set at the counter has been achieved, a contact will release electronic contactor r, thus bringing the motor to a standstill.

In accordance with the ASTM-regulations, 50 g of material to be ground of a particle size range of 590 - 11900 /um were fed. Grinding was terminated after 60 revolutions of the grinding shaft. The ground product was screened for 20 minutes on a stipulated US-screen of 0.074 mm of inside mesh width, i.e. a Retsch test screening machine. The grindability index as per Hardgrove was calculated in accordance with the following formula:

$$oH = 13 + 6.93\ W,$$

where W represents the passage through the 0.074 mm screen in g. For parallel tests a relative deviation of 2 - 3 % is generally considered tolerable for the individual grindability values.

The non-dimensional Hardgrove-Index oH can be converted into work index Wi of Bond via an empirical formula by R.W. Smith 12). A new equation has been developed on the basis of this formula yielding a specific energy requirement which corresponds to the values determined according to Zeisel in the following chapter. The new equation has been given below:

$$WH = \frac{435}{(oH)0.82}\ (kWh/t)$$

Hardgrove-Index oH and energy requirement WH ascertained on the basis of the Hardgrove-Index have been included in Table 3. The expenditure of energy WH stated there was related to a fineness of 12 % R 0.09, since WH was determined by approximation to the Zeisel values.

According to Zeisel the advantages offered by the Hardgrove-unit (small material quantity, simple evaluation, quick implementation) are facing considerable drawbacks which are that it is not possible to reach the grinding fineness degrees customary in the cement industry and to measure the expenditure of energy required for crushing. In view of the statements made under Section 3.1 above, this assertion of Zeisel is only conditionally valid. Zeisel has developed a new testing unit which retains the basic characteristics of the Hardgrove unit, but which features improvements.

The decisive difference is that the grinding bowl is not stationary but pivoted in the grindability tester. The grinding bowl is retained in zero position by a spring-type dynamometer arranged around the grinding bowl axle. The higher the grinding resistance of a sample, the stronger the spring extension. This extension is measured with the aid of a potentiometer versus time in counter units (ampere hours). These counter units can be converted into mkp or kWh via a calibration value.

The grindability tester is a product of Tonindustrie, Berlin. It has been time-tested for grindability tests in the cement industry. Technical details are described by H. Lehmann and U. Haese 2).

The mode of test implementation recommended by Zeisel 1) cannot be applied here, since preliminary tests carried out for limestones showed strong agglomeration phenomena and adhesion of the ground product to balls and grinding bowl. For this reason, an air separation and grinding process was imitated during the test. The feed size of 1.0 - 0.75 mm was maintained. From this fraction 25 g of material to be ground are fed to the grinding bowl and ground at intervals of 30 seconds (= 100 revolutions). The counter units are read off; afterwards, the overall material quantity is removed from the grinding bowl and split into final product and rejects at a cut point of 0.125 mm. The rejects are topped up to 25 g by prepared feed material to compensate the screened amount of final product and are again fed to the grinding bowl. From the final product the residues retained on the 0.09 mm screen (4900 M/cm2) are determined. This procedure is repeated until a final product quantity of approx. 25 g - corresponding to the feed - has been reached. The specific work index of the individual fractions is summarized and related to exactly 25 g. The results obtained with the aid of the grindability tester are multiplied by factor 3 for arriving at the expenditure of energy required for industrial-scale mills.

On the basis of Sample C (lime marl), the mode of evaluation of the test results has been shown below.

Example of calculation for Sample C for the changed mode of grindability testing as per Zeisel:

3.31 Test implementation

We would like to describe the test procedure on the basis of sample F, grinding to a size less than 0.315 mm, i.e. classification cut point 0.315 mm (Table 6 refers). For each grinding period the mill charge (feed: 700 cm3 of material, vibrated) amounts to 1208 g. Due to the screen analysis, the primary feed already contains 29.3 % by weight of grain less than 0.315 mm (Table 6), i.e. 1208 g x 0.293 = 354.0 g. 70 revolutions corresponding to 60 seconds of grinding period were selected for the first grinding period.

After the first grinding period the overall quantity of ground product was screened on a plane test screening machine at 0.315 mm and the residues were subjected to weighing. Screening yielded a residue of 678.2 g, i.e. 1208 g - 678.2 g = 529.8 g of ground product of a size less than 0.315 mm are available after the first period, of which 529.8 - 354 g = 175.8 g of a size less than 0.315 mm have been newly produced. A division of this newly produced quantity of ground product -0.315 mm by the number of revolutions selected will yield grindability coefficient G which

$$\text{in this case equals} \quad \frac{175.8 \text{ g}}{70 \text{ revolutions}} = 2.51 \text{ g/revolution.}$$

The quantity of undersize is substituted by primary feed and is again fed to the mill jointly with the outsize obtained from screening. This will ensure that the mill fed again contains 529.8 g . 0.293 = 155.2 g of grain of a size less than 0.315 mm.

As it is intended to implement the tests with an assumed circulating load of 250 %, grinding shall be done until

$\frac{1}{3.5}$ = 28.5 % of the feed (28.5 % of 1208 g = 345 g) of a size finer than 0.315 mm are at hand. Then, the number of revolutions for the second grinding period, i.e. 75.6 revolutions is

$$\text{calculated on the basis of} \quad \frac{3459 - 155.2}{2.51} \quad \text{in accordance with a}$$

grinding period of 64.8 seconds. This means that now a number of grinding periods is required that ensures that the amount of the grain -0.315 mm does not substantially deviate from 345 g over three grinding periods, i.e. a constant grindability coefficient G (g/revolution). This has been achieved for the 6th to 8th period of Table 6.

Grindability testing for all samples was done in accordarce with the pattern shown in Table 6 providing for intermediate screening at 0.2 mm and 0.1 mm. The grindability coefficients G were plotted in Table 7. This enabled a statement on the limits within which changes of the Bond work index, i.e. the specific work index (kWh/t) occur. Moreover, grinding including intermediate screening was carried out for samples A and F.

3.32 Calculation of work index Wi and specific work index W as per Bond

A precondition for calculation of work index Wi is to mix the fines from the last 3 grinding periods (6th to 8th period for Sample F) for having them subjected to a screen analysis on the Alpine air jet screen. The results of the fines and primary feed screen analysis are plotted in the logarithmic grain size distribution diagram according to Bond, which was done for Sample F included in Fig. 5. The D80-values of feed (F) and of ground product (P) are read from this diagram or calculated from distance x (cm) of cut point F and P from the ordinate on the basis of formula F and/or P = $\frac{10}{Vx}$. The results obtained have been summarized in Table 7. With values F and P ascertained such, mesh width Pi, at which screening was carried out, and grindability coefficient G, the work index can be calculated in the following manner:

$$Wi = \frac{4.45 \cdot 1.1 \cdot 1.3}{Pi^{0.23} \cdot G^{0.82} \cdot P^{-0.5} - F^{-0.5}} \quad ((kWh/t))$$

where:

Pi = mesh width at which screening was carried out (/um)
G = grindability coefficient (g/revolution)
F = D80-particle size of the feed material (/um)
P = D80-particle size of the final product (/um)
1.1 = factor for converting short tons in metric tons
1.3 = factor for conversion from wet to dry grinding

For all samples Wi has been specified in Table 7.

Specific work index W is calculated by plotting Wi in the basic Bond formula:

$$W = Wi \cdot \left(\frac{10}{VP} - \frac{10}{VFx} \right) \quad (kWh/t)$$

where:

Fx = D80 - value of feed for industrial-scale mill (/um)

The D80-values (F, Fx, P) can be gathered from the grain size distribution diagrams of the material as delivered (Fig.1) and the ground product (Fig. 2 - for reasons of space Bond diagrams have not been used here). These and specific work index W have been included in Table 7.

i.e. to the amount of final product which corresponds to the amount of feed material. The result are 55.76 mkp per 25 g of final product. Converted into kWh/t and multiplied by factor 3 this yields

$$55.76 \cdot \frac{3 \cdot 106}{25.102.3600} = 18.224 \text{ kWh/t of final product}$$

For calculating the fineness achieved at this specific work index, the overall amount of residues retained on the 0.09 mm screen is related to the total amount of material passing the 0.125 mm screen. A specific work index of 18.22 kWh/t at 13.19 % R 0.09 results. With the aid of the nomograph by W. Anselm 13) (Fig. 4) a conversion is made to the fineness of 12 % R 0.09 mm which is the usual standard in the cement industry. 18.9 kWh/t at 12 % R 0.09 are the result. A check test yielded 18.7 kWh/t at 12 % R 0.09.

All values ascertained with the aid of the grindability tester as per Zeisel have been included in Table 3 with their relevant fineness degrees.

3.3 B o n d t e s t 7-9)

The method developed by F.C. Bond serves for determining work index WI which characterizes the grindability features. The Bond test is a simplified model of a tube mill operating in closed circuit with a classifier. In this context, grinding is done in individual periods, in which the produced fines (finer than the reference fraction (0.1 mm, 0.2 mm, 0.315 mm) of the final product) are removed from the mill.

In the Bond ball mill of 305 mm inner diameter and 305 mm inner length with smooth wall and a ball charge defined as per Table 5, 700 cm3 of vibrated material 6 mesh (3.36 mm) at 70 rpm (n = 91 % ncrit) are ground by batches in closed circuit, dry at an assumed circulating load of 250 %. The ball charge specified by Bond (Table 5) was kept only approximately, as only balls of 40 mm, 30 mm, 25 mm, 19 mm and 13 mm diameter were available.

By way of tests carried out with ores, H.D. Wasmuth 3) found out that work index Wi is only slightly changed (+/- 0.5 ... - 2 %) compared to the origingal ball charge at a varying number of balls, however at an identical overall ball weight (20,125 kg). A value of 4.5 kp/l was determined for the bulk density of the ball charge. Thus, the laboratory mill filling degree equalled 21 %.

Similarly to the descriptions given under Section 3.2, the specific work index values W were converted to the standard fineness of 12 % R 0.09 with the aid of Fig. 4 and plotted in Table 8 as WB-values. To enable a comparison, this table likewise includes the grindability values as per Hardgrove and Zeisel which have been related to an identical fineness.

4. Discussion and comparison of the results

The scratch hardness has the greatest influence on the grindability after those characteristics mentioned in the beginning. As per Tables 1 and 8, the grindability is generally high in case of a great scratch hardness and vice versa. (Exception: Sample C). This dependency has also been established by U. Haese 14) for other raw materials.

Table 7 shows that work index Wi is fluctuating, which can be attributed to the material's primary particle size. The work index increases until the primary particle size has been reached and then drops again. Wim = 13.3 kWh/t which approximately corresponds to the value of 13.8 kWh/t tabulated by Bond 15) is obtained by ascertaining a mean work index Wim on the basis of all samples.

The following can be established by a comparison of the results of grindability testing with the aid of Table 8:

1. Hardgrove-Zeisel

Concerning grindability Hardgrove and Zeisel are at variance with each other. Presently leaving Sample D out of consideration, there exist deviations of the WH of WZ up to -23.8 % (Sample I) and + 6.3 % (Sample A + G). (WZ = 100 %). Sample D is loess, for which - due to its low grindability of only WZ = 5.4 kWh/t (100 %) - the percentage deviation of the WH is high, i.e. 38.8 %.

2. Hardgrove-Bond

Hardgrove and Bond are also at variance regarding their testing methods. There exist deviations of the WH of WB up to -22.3 % and +1.4 % at a maximum (WB = 100 %), subject to an omission of Sample D with a deviation of +5.6 %.

3. Zeisel-Bond

A comparison of these testing methods shows an almost identical situation. Again leaving Sample D out of consideration, the deviation is within a range from -18.2 % to +14.0 %.

In Figs. 6 - 8 the values obtained from the three testing procedures have been plotted versus each other, thus enabling a determination of the specific work index of one testing procedure on the basis of the other.

In concluding this Section it must be pointed out that all procedures described are not in agreement with each other.

The operating values included in Table 9 of cement plants A - I are representative values measured at industrial-scale mills over extended periods or calculated on the basis of the relevant documentation. All specific work index values in this paper relate to a fineness of 12 % R 0.09 mm. Samples A and B are ground in the relevant cement plant as a mixture of a ratio A : B. The specific work index values have been converted accordingly for this mixture: WH = 10.53 kWh/t, WZ = 10.15 kWh/t, WB = 10.43 kWh/t.

The operating values have been plotted in Table 9 jointly with the mill dimensions and the type of industrial-scale grinding plants. As to Sample D, no data on the industrial-scale mill were at hand. Table 9 also includes the design data of mills which have been laid out for grindability testing as per Zeisel.

Actually, it was intended to compare the projected mills on the basis of the grindability values as per Zeisel, Bond and Hardgrove with the industrial-scale mills. As it is the Zeisel test that is mainly implemented in the cement industry, this test was carried out first.

The design data of the projected mills in Table 9 have been obtained independent of the values of the industrial-scale mills. Only the feed size as per Fig. 1 was considered for the design. For the grindability values according to Zeisel WZ, a safety margin of 5 - 10 % was imposed for all mills. The Sample H value was given an additional tolerance on account of a moisture content of 6.2 %.

Due to the high air requirement the trunnion bearings in the air-swept grinding installations have been amply dimensioned and only permit a ball charge of 26 %. Therefore, the mill tubes of the air-swept grinding installations are the longest. The closed-circuit grinding plants with bucket elevator have been filled with balls to 28 % and projected with a shorter mill tube for all cases mentioned in Table 9. The mill tubes of Tandem grinding installations shall be designed with shortest mill tubes and in most of the cases also with smaller diameter, as intensive preliminary crushing in the impact hammer mill permits tube mill designs smaller by 20 % and more 16).

The tests have been carried out in the test center of KHD-Industrieanlagen AG, Humboldt-Wedag, Cologne. The authors would like to thank the gentlemen Auel and J. Gruener for the careful implementation of the tests.

LAVAL LAB INC.
2895 Bergman
Laval, Qc, Canada H7L 3Y5
(514) 688-5545

Designation: C 204 – 84

AMERICAN SOCIETY FOR TESTING AND MATERIALS
1916 Race St., Philadelphia, Pa. 19103
Reprinted from the Annual Book of ASTM Standards, Copyright ASTM
If not listed in the current combined index, will appear in the next edition.

Standard Test Method for FINENESS OF PORTLAND CEMENT BY AIR PERMEABILITY APPARATUS[1]

This standard is issued under the fixed designation C 204; the number immediately following the designation indicates the year of original adoption or, in the case of revision, the year of last revision. A number in parentheses indicates the year of last reapproval. A superscript epsilon (ε) indicates an editorial change since the last revision or reapproval.

1. Scope

1.1 This test method covers determination of the fineness of portland cement, using the Blaine air permeability apparatus, in terms of the specific surface expressed as total surface area in square centimetres per gram, or square metres per kilogram, of cement. Although the test method may be, and has been, used for the determination of the measures of fineness of various other materials, it should be understood that, in general, relative rather than absolute fineness values are obtained.

1.2 The values stated in acceptable metric units are to be regarded as the standard.

1.3 *This standard may involve hazardous materials, operations, and equipment. This standard does not purport to address all of the safety problems associated with its use. It is the responsibility of whoever uses this standard to consult and establish appropriate safety and health practices and determine the applicability of regulatory limitations prior to use.*

2. Applicable Documents

2.1 *ASTM Standard:*

C670 Practice for Preparing Precision Statements for Test Methods for Construction Materials[2]

2.2 *Other Documents:*

UU-P-236 Federal Specification for Paper; Filtering[3]

No. 114 National Bureau of Standards Standard Reference Material[4]

3. Apparatus

3.1 *Nature of Apparatus*—The Blaine air permeability apparatus consists essentially of a means of drawing a definite quantity of air through a prepared bed of cement of definite porosity. The number and size of the pores in a prepared bed of definite porosity is a function of the size of the particles and determines the rate of airflow through the bed. The apparatus, illustrated in Fig. 1, shall consist specifically of the parts described in 3.2 to 3.8.

3.2 *Permeability Cell*—The permeability cell shall consist of a rigid cylinder 12.70 ± 0.10 mm in inside diameter, constructed of austenitic stainless steel. The interior of the cell shall have a finish of 0.81 μm (32 μin.). The top of the cell shall be at right angles to the principal axis of the cell. The lower portion of the cell must be able to form an airtight fit with the upper end of the manometer, so that there is no air leakage between the contacting surfaces. A ledge ½ to 1 mm in width shall be an integral part of the cell or be firmly fixed in the cell 55 ± 10 mm from the top of the cell for support of the perforated metal disk. The top of the permeability cell shall be fitted with a protruding collar to facilitate the removal of the cell from the manometer.

NOTE 1—AISI Type 303 stainless steel (UNS designation S30300) has been found to be suitable for the construction of the permeability cell and the plunger.

---

[1] This test method is under the jurisdiction of ASTM Committee C-1 on Cement and is the direct responsibility of Subcommittee C01.25 on Fineness.

Current edition approved Feb. 24, 1984. Published April 1984. Originally published as C 204 – 46 T. Last previous edition C 204 – 81[ε1].

[2] *Annual Book of ASTM Standards*, Vol 04.01.

[3] Available from Naval Publications and Forms Center, 5801 Tabor Ave., Philadelphia, PA 19120.

[4] Available from National Bureau of Standards, Office of Standard Reference Materials, Chemistry Bldg., Washington, D.C. 20234.

3.3 *Disk*—The disk shall be constructed of noncorroding metal and shall be 0.9 ± 0.1 mm in thickness, perforated with 30 to 40 holes 1 mm in diameter equally distributed over its area. The disk shall fit the inside of the cell snugly. The center portion of one side of the disk shall be marked or inscribed in a legible manner so as to permit the operator always to place that side downwards when inserting it into the cell. The marking or inscription shall not extend into any of the holes, nor touch their peripheries, nor extend into that area of the disk that rests on the cell ledge.

3.4 *Plunger*—The plunger shall be constructed of austenitic stainless steel and shall fit into the cell with a clearance of not more than 0.1 mm. The bottom of the plunger shall sharply meet the lateral surfaces and shall be at right angles to the principal axis. An air vent shall be provided by means of a flat 3.0 ± 0.3 mm wide on one side of the plunger. The top of the plunger shall be provided with a collar such that when the plunger is placed in the cell and the collar brought in contact with the top of the cell, the distance between the bottom of the plunger and the top of the perforated disk shall be 15 ± 1 mm.

3.5 *Filter Paper*—The filter paper shall be medium retentive, corresponding to Type 1, Grade B, as prescribed in Federal Specification for Paper; Filtering (UU-P-236). The filter paper disks shall be circular, with smooth edges, and shall have the same diameter (Note 2) as the inside of the cell.

NOTE 2—Filter paper disks that are too small may leave part of the sample adhering to the inner wall of the cell above the top disk. When too large in diameter, the disks have a tendency to buckle and cause erratic results.

3.6 *Manometer*—The U-tube manometer shall be constructed according to the design indicated in Fig. 1, using nominal 9-mm outside diameter, standard-wall, glass tubing. The top of one arm of the manometer shall form an airtight connection with the permeability cell. The manometer arm connected to the permeability cell shall have a line etched around the tube at 125 to 145 mm below the top side outlet and also others at distances of 15 ± 1 mm, 70 ± 1 mm, and 110 ± 1 mm above that line. A side outlet shall be provided at 250 to 305 mm above the bottom of the manometer for use in the evacuation of the manometer arm connected to the permeability cell. A positive airtight valve or clamp shall be provided on the side outlet not more than 50 mm from the manometer arm. The manometer shall be mounted firmly and in such a manner that the arms are vertical.

3.7 *Manometer Liquid*—The manometer shall be filled to the midpoint with a nonvolatile, nonhygroscopic liquid of low viscosity and density, such as dibutyl phthalate (dibutyl 1,2-benzene-dicarboxylate)or a light grade of mineral oil.

3.8 *Timer*—The timer shall have a positive starting and stopping mechanism and shall be capable of being read to the nearest 0.5 s or less. The timer shall be accurate to 0.5 s or less for time intervals up to 60 s, and to 1 % or less for time intervals of 60 to 300 s.

4. Calibration of Apparatus

4.1 *Sample*—The calibration of the air permeability apparatus shall be made using the current lot of NBS Standard Reference Material No. 114. The sample shall be at room temperature when tested.

4.2 *Bulk Volume of Compacted Bed of Powder*—Determine the bulk volume of the compacted bed of powder by the mercury displacement method as follows:

4.2.1 Place two filter paper disks in the permeability cell, pressing down the edges, using a rod having a diameter slightly smaller than that of the cell, until the filter disks are flat on the perforated metal disk; then fill the cell with mercury, ACS reagent grade or better, removing any air bubbles adhering to the wall of the cell. Use tongs when handling the cell. If the cell is made of material that will amalgamate with mercury, the interior of the cell shall be protected by a very thin film of oil just prior to adding the mercury. Level the mercury with the top of the cell by lightly pressing a small glass plate against the mercury surface until the glass is flush to the surface of the mercury and rim of the cell, being sure that no bubble or void exists between the mercury surface and the glass plate. Remove the mercury from the cell, weigh, and record the weight of the mercury. Remove one of the filter disks from the cell. Using a trial quantity of 2.80 g of cement (Note 3) compress the cement (Note 4) in accordance with 4.5 with one filter disk above and one below the sample. Into the unfilled

space at the top of the cell, add mercury, remove entrapped air, and level off the top as before. Remove the mercury from the cell, weigh, and record the weight of the mercury.

4.2.2 Calculate the bulk volume occupied by the cement to the nearest 0.005 cm^3 as follows:

$$V = (W_A - W_B)/D \quad (1)$$

where:

V = bulk volume of cement, cm^3,
W_A = grams of mercury required to fill the cell, no cement being in the cell,
W_B = grams of mercury required to fill the portion of the cell not occupied by the prepared bed of cement in the cell, and
D = density of mercury at the temperature of test, g/cm^3 (see Table 1).

4.2.3 Make at least two determinations of bulk volume of cement, using separate compactions for each determination. The bulk volume value used for subsequent calculations shall be the average of two values agreeing within ± 0.005 cm^3. Note the temperature in the vicinity of the cell and record at the beginning and end of the determination.

NOTE 3—It is not necessary to use the standard sample for the bulk volume determination.

NOTE 4—The prepared bed of cement shall be firm. If too loose or if the cement cannot be compressed to the desired volume, adjust the trial quantity of cement used.

4.3 *Preparation of Sample*—Enclose the contents of a vial of the standard cement sample in a jar, approximately 120 cm^3 (4 oz), and shake vigorously for 2 min to fluff the cement and break up lumps or agglomerates. Allow the jar to stand unopened for a further 2 min, then remove the lid and stir gently to distribute throughout the sample the fine fraction that has settled on the surface after fluffing.

4.4 *Weight of Sample*—The weight of the standard sample used for the calibration test shall be that required to produce a bed of cement having a porosity of 0.500 ± 0.005, and shall be calculated as follows:

$$W = \rho V(1 - \epsilon) \quad (2)$$

where:

W = grams of sample required,
ρ = density of test sample (for portland cement a value of 3.15 Mg/m^3 or 3.15 g/cm^3 shall be used),
V = bulk volume of bed of cement, cm^3, as determined in accordance with 4.2, and
ϵ = desired porosity of bed of cement (0.500 ± 0.005) (Note 5).

NOTE 5—The porosity is the ratio of volume of voids in a bed of cement to the total or bulk volume of the bed, V.

4.5 *Preparation of Bed of Cement*—Seat the perforated disk on the ledge in the permeability cell, inscribed or marked face down. Place a filter paper disk on the metal disk and press the edges down with a rod having a diameter slightly smaller than that of the cell. Weigh to the nearest 0.001 g the quantity of cement determined in accordance with 4.4 and place in the cell. Tap the side of the cell lightly in order to level the bed of cement. Place a filter paper disk on top of the cement and compress the cement with the plunger until the plunger collar is in contact with the top of the cell. Slowly withdraw the plunger a short distance, rotate about 90°, repress, and then slowly withdraw. Use of fresh disks is required for each determination.

4.6 *Permeability Test:*

4.6.1 Attach the permeability cell to the manometer tube, making certain that an airtight connection is obtained (Note 6) and taking care not to jar or disturb the prepared bed of cement.

4.6.2 Slowly evacuate the air in the one arm of the manometer U-tube until the liquid reaches the top mark, and then close the valve tightly. Start the timer when the bottom of the meniscus of the manometer liquid reaches the second (next to the top) mark and stop when the bottom of the meniscus of liquid reaches the third (next to the bottom) mark. Note the time interval measured and record in seconds. Note the temperature of test and record in degrees Celsius.

4.6.3 In the calibration of the instrument, make at least three determinations of the time of flow on each of three separately prepared beds of the standard sample (Note 7). The calibration shall be made by the same operator who makes the fineness determination.

NOTE 6—A little stopcock grease should be applied to the standard taper connection. The efficiency of the connection can be determined by attaching the cell to the manometer, stoppering it, partially evacuating the one arm of the manometer, then closing the valve. Any continuous drop in pressure indicates a leak in the system.

NOTE 7—The sample may be refluffed and reused for preparation of the test bed, provided that it is kept dry and all tests are made within 4 h of the opening of the sample.

4.7 *Recalibration*—The apparatus shall be recalibrated (Note 8):

4.7.1 At periodic intervals to correct for possible wear on the plunger or permeability cell,

4.7.2 If any loss in the manometer fluid occurs, or

4.7.3 If a change is made in the type or quality of the filter paper used for the tests.

NOTE 8—It is suggested that a secondary sample be prepared and used as a fineness standard for the check determinations of the instrument between regular calibrations with the standard cement sample.

5. Procedure

5.1 *Temperature of Cement*—The cement sample shall be at room temperature when tested.

5.2 *Size of Test Sample*—The weight of sample (Note 9) used for the test shall be the same as that used in the calibration test on the standard sample, except that when determining the fineness of Type III cements, or other types of fine-ground cements whose bulk for this weight is so great that ordinary thumb pressure will not cause the plunger collar to contact the top of the cell, the weight of sample shall be that required to produce a test bed having a porosity of 0.530 ± 0.005.

NOTE 9—When this method is used for materials other than portland cement, or if for a portland cement sample one of the required porosities given in 5.2 cannot be attained, the weight of sample shall be adjusted so that a firm, hard bed is produced by the compacting process. In no case, however, should more than thumb pressure be used to secure the proper bed, nor should such thumb pressure be used that the plunger "rebounds" from the cell top when pressure is removed.

5.3 *Preparation of Bed of Cement*—Prepare the test bed of cement in accordance with the method described in 4.5.

5.4 *Permeability Tests*—Make the permeability tests in accordance with the method described in 4.6, except that only one time-of-flow determination need be made on each bed.

6. Calculation

6.1 Calculate the specific surface values in accordance with the following equations:

$$S = \frac{S_s\sqrt{T}}{\sqrt{T_s}} \tag{3}$$

$$S = \frac{S_s\sqrt{\eta}\sqrt{T}}{\sqrt{T_s}\sqrt{\eta_s}} \tag{4}$$

$$S = \frac{S_s(b-\epsilon_s)\sqrt{\epsilon^3}\sqrt{T}}{\sqrt{\epsilon_s^3}\sqrt{T_s}(b-\epsilon)} \tag{5}$$

$$S = \frac{S_s(b-\epsilon_s)\sqrt{\epsilon^3}\sqrt{\eta_s}\sqrt{T}}{\sqrt{\epsilon_s^3}\sqrt{T_s}\sqrt{\eta}(b-\epsilon)} \tag{6}$$

$$S = \frac{S_{sd}(b-\epsilon_s)\sqrt{\epsilon^3}\sqrt{T}}{\rho(b-\epsilon)\sqrt{\epsilon_s^3}\sqrt{T_s}} \tag{7}$$

$$S = \frac{S_{sd}(b-\epsilon_s)\sqrt{\eta_s}\sqrt{\epsilon^3}\sqrt{T}}{\rho(b-\epsilon)\sqrt{\epsilon_s^3}\sqrt{T_s}\sqrt{\eta}} \tag{8}$$

where:

S = specific surface of the test sample, cm²/g,

S_s = specific surface of the standard sample used in calibration of the apparatus, cm²/g,

T = measured time interval, s, of manometer drop for test sample (Note 10),

T_s = measured time interval, s, of manometer drop for standard sample used in calibration of the apparatus (Note 10),

η = viscosity of air, poises, at the temperature of test of the test sample (Note 10),

η_s = viscosity of air, poises, at the temperature of test of the standard sample used in calibration of the apparatus (Note 10),

ϵ = porosity of prepared bed of test sample (Note 10),

ϵ_s = porosity of prepared bed of standard sample used in calibration of apparatus (Note 10),

ρ = density of test sample (for portland cement a value of 3.15 Mg/m³ or 3.15 g/cm³ shall be used),

ρ_s = density of standard sample used in calibration of apparatus (assumed to be 3.15 Mg/m³ or 3.15 g/cm³), and

b = a constant (for portland cement a value of 0.9 shall be used).

NOTE 10—Values for $\sqrt{\eta}$, $\sqrt{\epsilon^3}$, and $\sqrt{T}$ may be taken from Tables 1, 2, and 3, respectively.

Equations 3 and 4 shall be used in calculations of fineness of portland cements compacted to the same porosity as the standard fineness sample. Equation 3 is used if the temperature of test of the test sample is within ±3°C of the temperature of calibration test, and Eq 4 is used if the temperature of test of the test sample is outside of this range.

Equations 5 and 6 shall be used in calculation of fineness of portland cements compacted to some porosity other than that of the standard fineness sample used in the calibration test. Equation 5 is used if the temperature of test of the test sample is within ±3°C of the temperature of calibration test of the standard fineness sample, and Eq 6 is used if the temperature of test of the test sample is outside of this range.

Equations 7 and 8 shall be used in calculation of fineness of materials other than portland cement. Equa-

tion 7 shall be used when the temperature of test of the test sample is within ±3°C of the temperature of calibration test, and Eq 8 is used if the temperature of test of the test sample is outside of this range.

It is recommended that values of b be determined on no less than three samples of the material in question. Test each sample at a minimum of four different porosities over a porosity range of at least 0.06. Correlation coefficients should exceed 0.9970 for the correlation of $\sqrt{\epsilon^3}T$ versus ϵ on each sample tested (see Appendix X1).

6.2 To calculate the specific surface values in square metres per kilogram, multiply the surface area in cm^2/g by the factor of 0.1.

6.3 Round values in cm^2/g to the nearest 10 units (in m^2/kg to the nearest unit). Example: 3447 cm^2/g is rounded to 3450 cm^2/g or 345 m^2/kg.

7. Report

7.1 For portland cements and portland cement-based materials, report results on a single determination on a single bed.

7.2 For very high fineness materials with long time intervals, report the average fineness value of two permeability tests, provided that the two agree within 2 % of each other. If they do not agree, discard the values and repeat the test (Note 11) until two values so agreeing are obtained.

NOTE 11—Lack of agreement indicates a need for checks of procedure and apparatus. Care shall be exercised in the preparation of the test beds, and precautions shall be taken to ensure an airtight connection between the permeability cell and the manometer arm. See also the "Manual of Cement Testing."[2]

8. Precision

8.1 *Single-Operator Precision*—The single-operator coefficient of variation has been found to be 1.2 % (Note 12). Therefore, results of two properly conducted tests, by the same operator, on the same sample, should not differ by more than 3.4 % (Note 12) of their average.

8.2 *Multilaboratory Precision*—The multilaboratory coefficient of variation has been found to be 2.1 % (Note 12). Therefore, results of two different laboratories on identical samples of a material should not differ from each other by more than 6.0 % (Note 12) of their average.

NOTE 12—These numbers represent, respectively, the 1S % and D2S % limits as described in Practice C 670.

TABLE 1 Density of Mercury, Viscosity of Air (η), and $\sqrt{\eta}$ at Given Temperatures

Room Temperature, °C	Density of Mercury, g/cm³	Viscosity of Air, η	$\sqrt{\eta}$
16	13.56	0.0001788	0.01337
18	13.55	0.0001798	0.01341
20	13.55	0.0001808	0.01345
22	13.54	0.0001818	0.01348
24	13.54	0.0001828	0.01352
26	13.53	0.0001837	0.01355
28	13.53	0.0001847	0.01359
30	13.52	0.0001857	0.01363
32	13.52	0.0001867	0.01366
34	13.51	0.0001876	0.01370

TABLE 2 Values for Porosity of Cement Bed

Porosity of Bed, ϵ	$\sqrt{\epsilon^3}$
0.495	0.348
0.496	0.349
0.497	0.350
0.498	0.351
0.499	0.352
0.500	0.354
0.501	0.355
0.502	0.356
0.503	0.357
0.504	0.358
0.505	0.359
0.506	0.360
0.507	0.361
0.508	0.362
0.509	0.363
0.510	0.364
0.525	0.380
0.526	0.381
0.527	0.383
0.528	0.384
0.529	0.385
0.530	0.386
0.531	0.387
0.532	0.388
0.533	0.389
0.534	0.390
0.535	0.391

TABLE 3 Time of Airflow

T = time of airflow in seconds; $\sqrt{T}$ = the factor for use in the equations

T	$\sqrt{T}$	*T*	$\sqrt{T}$	*T*	$\sqrt{T}$	*T*	$\sqrt{T}$	*T*	$\sqrt{T}$	*T*	$\sqrt{T}$
26	5.10	51	7.14	76	8.72	101	10.05	151	12.29	201	14.18
26½	5.15	51½	7.18	76½	8.75	102	10.10	152	12.33	202	14.21
27	5.20	52	7.21	77	8.77	103	10.15	153	12.37	203	14.25
27½	5.24	52½	7.25	77½	8.80	104	10.20	154	12.41	204	14.28
28	5.29	53	7.28	78	8.83	105	10.25	155	12.45	205	14.32
28½	5.34	53½	7.31	78½	8.86	106	10.30	156	12.49	206	14.35
29	5.39	54	7.35	79	8.89	107	10.34	157	12.53	207	14.39
29½	5.43	54½	7.38	79½	8.92	108	10.39	158	12.57	208	14.42
30	5.48	55	7.42	80	8.94	109	10.44	159	12.61	209	14.46
30½	5.52	55½	7.45	80½	8.97	110	10.49	160	12.65	210	14.49
31	5.57	56	7.48	81	9.00	111	10.54	161	12.69	211	14.53
31½	5.61	56½	7.52	81½	9.03	112	10.58	162	12.73	212	14.56
32	5.66	57	7.55	82	9.06	113	10.63	163	12.77	213	14.59
32½	5.70	57½	7.58	82½	9.08	114	10.68	164	12.81	214	14.63
33	5.74	58	7.62	83	9.11	115	10.72	165	12.85	215	14.66
33½	5.79	58½	7.65	83½	9.14	116	10.77	166	12.88	216	14.70
34	5.83	59	7.68	84	9.17	117	10.82	167	12.92	217	14.73
34½	5.87	59½	7.71	84½	9.19	118	10.86	168	12.96	218	14.76
35	5.92	60	7.75	85	9.22	119	10.91	169	13.00	219	14.80
35½	5.96	60½	7.78	85½	9.25	120	10.95	170	13.04	220	14.83
36	6.00	61	7.81	86	9.27	121	11.00	171	13.08	222	14.90
36½	6.04	61½	7.84	86½	9.30	122	11.05	172	13.11	224	14.97
37	6.08	62	7.87	87	9.33	123	11.09	173	13.15	226	15.03
37½	6.12	62½	7.91	87½	9.35	124	11.14	174	13.19	228	15.10
38	6.16	63	7.94	88	9.38	125	11.18	175	13.23	230	15.17
38½	6.20	63½	7.97	88½	9.41	126	11.22	176	13.27	232	15.23
39	6.24	64	8.00	89	9.43	127	11.27	177	13.30	234	15.30
39½	6.28	64½	8.03	89½	9.46	128	11.31	178	13.34	236	15.36
40	6.32	65	8.06	90	9.49	129	11.36	179	13.38	238	15.43
40½	6.36	65½	8.09	90½	9.51	130	11.40	180	13.42	240	15.49
41	6.40	66	8.12	91	9.54	131	11.45	181	13.45	242	15.56
41½	6.44	66½	8.15	91½	9.57	132	11.49	182	13.49	244	15.62
42	6.48	67	8.19	92	9.59	133	11.53	183	13.53	246	15.68
42½	6.52	67½	8.22	92½	9.62	134	11.58	184	13.56	248	15.75
43	6.56	68	8.25	93	9.64	135	11.62	185	13.60	250	15.81
43½	6.60	68½	8.28	93½	9.67	136	11.66	186	13.64	252	15.87
44	6.63	69	8.31	94	9.70	137	11.70	187	13.67	254	15.94
44½	6.67	69½	8.34	94½	9.72	138	11.75	188	13.71	256	16.00
45	6.71	70	8.37	95	9.75	139	11.79	189	13.75	258	16.06
45½	6.75	70½	8.40	95½	9.77	140	11.83	190	13.78	260	16.12
46	6.78	71	8.43	96	9.80	141	11.87	191	13.82	262	16.19
46½	6.82	71½	8.46	96½	9.82	142	11.92	192	13.86	264	16.25
47	6.86	72	8.49	97	9.85	143	11.96	193	13.89	266	16.31
47½	6.89	72½	8.51	97½	9.87	144	12.00	194	13.93	268	16.37
48	6.93	73	8.54	98	9.90	145	12.04	195	13.96	270	16.43
48½	6.96	73½	8.57	98½	9.92	146	12.08	196	14.00	272	16.49
49	7.00	74	8.60	99	9.95	147	12.12	197	14.04	274	16.55
49½	7.04	74½	8.63	99½	9.97	148	12.17	198	14.07	276	16.61
50	7.07	75	8.66	100	10.00	149	12.21	199	14.11	278	16.67
50½	7.11	75½	8.69	100½	10.02	150	12.25	200	14.14	280	16.73

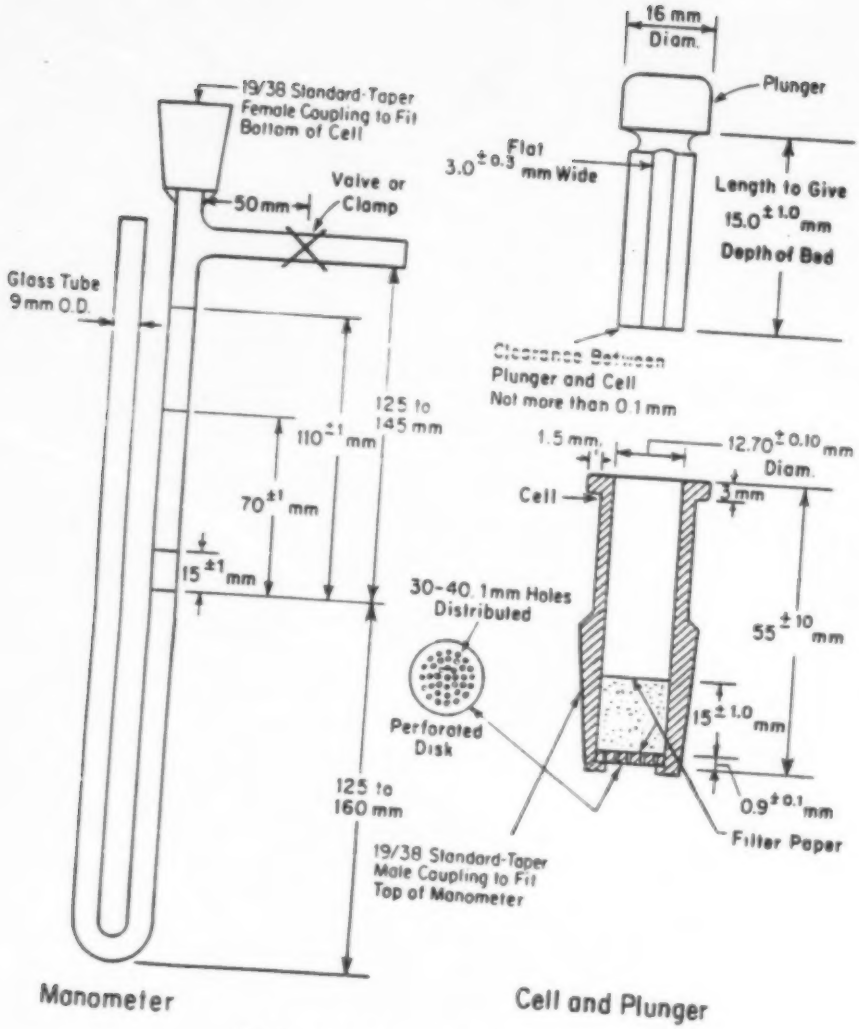


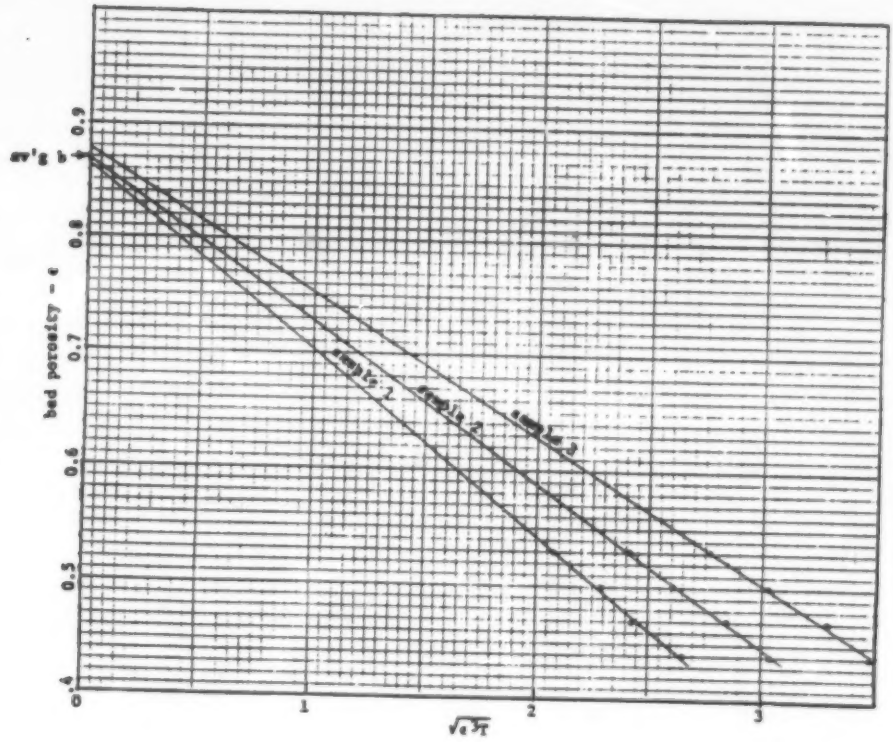
FIG. 1 Blaine Air Permeability Apparatus

The American Society for Testing and Materials takes no position respecting the validity of any patent rights asserted in connection with any item mentioned in this standard. Users of this standard are expressly advised that determination of the validity of any such patent rights, and the risk of infringement of such rights, are entirely their own responsibility.

This standard is subject to revision at any time by the responsible technical committee and must be reviewed every five years and if not revised, either reapproved or withdrawn. Your comments are invited either for revision of this standard or for additional standards and should be addressed to ASTM Headquarters. Your comments will receive careful consideration at a meeting of the responsible technical committee, which you may attend. If you feel that your comments have not received a fair hearing you should make your views known to the ASTM Committee on Standards, 1916 Race St., Philadelphia, Pa. 19103.

APPENDIX

(Nonmandatory Information)



	ε	W	T	$\sqrt{\varepsilon^3 T}$
		Sample 1		
	0.530	2.350	29.0	2.078
	0.500	2.500	42.0	2.291
	0.470	2.650	57.5	2.443
	0.440	2.800	82.5	2.651
		Sample 2		
	0.530	2.350	39.0	2.410
	0.500	2.500	55.5	2.610
	0.470	2.650	79.0	2.846
	0.440	2.800	108.5	3.040
		Sample 3		
	0.530	2.350	51.5	2.769
	0.500	2.500	73.0	3.021
	0.470	2.650	104.0	3.286
	0.440	2.800	141.5	3.472

Material: Silica flour

ρ = density of test sample = 2.65 g/cm³
V = bulk volume of sample bed = 1.887 cm³
ε = desired porosity of test
W = grams of sample required = $\rho V(1 - \varepsilon)$
T = measured test time interval, seconds

Computed values of b by linear regression:

Sample 1 b = 0.863 (correlation coefficient = 0.9980)
Sample 2 b = 0.869 (correlation coefficient = 0.9993)
Sample 3 b = 0.879 (correlation coefficient = 0.9973)
Average b = 0.870

FIG. X1.1 Illustrative Method for the Determination of the Value for the Constant b (for use in fineness calculations of materials other than portland cement)